ASX Announcement

25 June 2014

ANNUAL MINERAL RESOURCES AND ORE RESERVES STATEMENT

Evolution Mining (ASX: EVN) advises that it has completed the annual update to its Mineral Resource and Ore Reserve estimates, current as at 31 December 2013.

Group Ore Reserves are now estimated at 52 million tonnes at 1.6g/t AuEq for 2.62 million ounces gold equivalent, a decrease of approximately 0.95 million ounces compared with the estimate at 31 December 2012 of 75.9 million tonnes at 1.5g/t AuEq for 3.57 million ounces gold equivalent. The change is predominantly a result of depletion by mining and using more conservative economic parameters and modelling techniques. The main changes were at Edna May and Pajingo where the resource category of mineralisation peripheral to the main ore bodies has been downgraded pending additional drilling and geological modelling (i.e. material previously included in Indicated Resources was downgraded to Inferred and therefore did not qualify for inclusion in the Ore Reserve estimate).

Current infill drilling programmes to convert Mineral Resources to Ore Reserves are focussed on increasing the confidence in each resource category by using drill spacing appropriate to the lode geometry and complexity. Evolution is confident that significant Mineral Resources and Ore Reserves will be added to the Group inventory as a result of this work.

The Pajingo and Cracow underground mines have a long history of reserve replacement. Evolution remains confident in the long term future of these mines given the current Mineral Resource inventory, the nature of the geology and the historic high conversion rate of Mineral Resources to Ore Reserves. Both mines continue to maintain robust 5 year life-of-mine plans, which is typical of the way these mines have operated for many years.

Group Mineral Resources are now estimated at 107 million tonnes at 1.7g/t AuEq for 5.70 million ounces gold equivalent, a decrease of approximately 1.99 million ounces compared with the estimate at 31 December 2012 of 148 million tonnes at 1.6g/t AuEq for 7.69 million ounces gold equivalent. The change is predominantly a result of depletion by mining and also a more stringent application of economic constraints appropriate to the type of individual orebody concerned. The updated Mineral Resources are now constrained by open pit or underground stope shapes (as relevant) constructed according to a long-term gold price assumption of A$1,800 per ounce. Previously, most of the Group Mineral Resources were not constrained. This change has impacted Edna May and Mt Carlton. The reportable Mineral Resource has decreased by 945koz gold equivalent as a result of this change however it is important to note that the underlying potential to convert Mineral Resources to Ore Reserves with additional drilling and economic studies has not changed materially. The update also reflects an improved geological understanding of the controls on mineralisation within new areas mined over the update period.

Since mid-2013 the strategic focus of Evolution's exploration efforts has been to build a platform for step-change transformational discovery. Recent exploration has focused primarily on building 4D models supported by 2D and 3D seismic, to interrogate large historical databases to improve area selection and the location of specific drill targets. The objective is to increase the likelihood of exploration success, shorten the timeframe, and decrease the cost and number of drill-holes to make new discoveries. This process will bring forward new Inferred Resources for ultimate conversion to Ore Reserves. With this work now coming to fruition, a significant increase in the amount of drilling is expected in FY15. Current plans estimate that over 80,000m will be drilled at Evolution's exploration properties in FY15 – two and half times the amount drilled in FY14 (approximately 31,875m).

Commenting on the updated Mineral Resource and Ore Reserve inventory, Evolution Executive Chairman, Jake Klein, said:

> "Evolution has taken a more conservative view of its estimation of resources and reserves which is appropriate for the current gold price environment. This approach has meant that some mineralisation previously included in resource and reserve estimates has now been excluded largely due to more conservative parameters being applied to the modelling approach. We do not anticipate these changes to have any effect on our production profile. We remain confident in the long term potential of our mines and are excited about the exploration work focused on transformational discovery that is currently underway."

Mineral Resources are reported inclusive of Ore Reserves and includes all exploration and resource definition drilling information up to 31 December 2013 and has been depleted for mining to 31 December 2013. Commodity price assumptions used to estimate Mineral Resources and Ore Reserves are similar to those used previously:

- Gold: A$1,350/oz
- Silver: A$22.00/oz
- Copper: A$3.00/lb

The Mineral Resources and Ore Reserves statement attached to this announcement has been prepared in accordance with the JORC Code 2012 for all projects other than Twin Hills. The Twin Hills Mineral Resource was first disclosed under JORC Code 2004 requirements and has not been updated to JORC Code 2012 requirements as it is not currently classified as a material mining project.

Group Mineral Resource and Ore Reserve summaries are tabulated on the following pages. Information for the individual projects including a Material Information Summary pursuant to ASX Listing Rules 5.8 and 5.9 and the Assessment and Reporting Criteria in accordance with JORC Code 2012 requirements are also included.

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About Evolution Mining

Evolution Mining is a leading, growth-focused Australian gold miner. The Company operates five wholly-owned Australian mines – Cracow, Mt Carlton, Mt Rawdon and Pajingo in Queensland and Edna May in Western Australia.

Group production in FY14 is forecast to be between 400,000 – 450,000 ounces gold equivalent with cash operating costs expected to be in the range of A$770 – A$820 per ounce. The additional costs of royalties, deferred open pit stripping, rehabilitation, sustaining capital and corporate overheads add approximately A$310/oz providing for Group All-in Sustaining Costs of A$1,080/oz to A$1,130/oz.

December 2013 Group Ore Reserve Statement

Gold			Proved			Probable			Total Reserve			
Project	**Type**	**Cut-Off**	**Tonnes (Mt)**	**Gold Grade (g/t)**	**Gold Metal (koz)**	**Tonnes (Mt)**	**Gold Grade (g/t)**	**Gold Metal (koz)**	**Tonnes (Mt)**	**Gold Grade (g/t)**	**Gold Metal (koz)**	**Competent Person**
Cracow[1]	Underground	3.5	0.36	7.3	83	1.00	5.5	176	1.36	5.9	260	1
Pajingo[1]	Underground	3.3	0.18	7.1	40	0.60	6.0	114	0.77	6.2	155	1
Edna May[1]	Open-Pit	0.5	-	-	-	11.35	1.1	402	11.35	1.1	402	2
Mt Carlton[1]	Open-Pit	0.9	0.19	1.6	10	7.11	3.0	695	7.30	3.0	705	3
Mt Rawdon[1]	Open-Pit	0.3	0.76	0.5	12	29.80	0.9	850	30.56	0.9	862	3
Total			**1.48**	**3.1**	**146**	**49.86**	**1.4**	**2,237**	**51.34**	**1.5**	**2,383**	

Silver			Proved			Probable			Total Reserve			
Project	**Type**	**Cut-Off**	**Tonnes (Mt)**	**Silver Grade (g/t)**	**Silver Metal (koz)**	**Tonnes (Mt)**	**Silver Grade (g/t)**	**Silver Metal (koz)**	**Tonnes (Mt)**	**Silver Grade (g/t)**	**Silver Metal (koz)**	
Mt Carlton[1]	Open-Pit	*	0.45	50	722	7.52	37	8,841	7.97	38	9,563	3
Total			**0.45**	**50**	**722**	**7.52**	**37**	**8,841**	**7.97**	**38**	**9,563**	

Copper			Proved			Probable			Total Reserve			
Project	**Type**	**Cut-Off**	**Tonnes (Mt)**	**Copper Grade (%)**	**Copper Metal (kt)**	**Tonnes (Mt)**	**Copper Grade (%)**	**Copper Metal (kt)**	**Tonnes (Mt)**	**Copper Grade (%)**	**Copper Metal (kt)**	
Mt Carlton[1]	Open-Pit	*	0.45	0.3	1.6	7.52	0.28	21.3	7.97	0.3	23	3
Total			**0.45**	**0.3**	**1.6**	**7.52**	**0.28**	**21.3**	**7.97**	**0.3**	**23**	

Gold Equivalence	Tonnes (Mt)	Gold Equiv. Grade (g/t)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Grade (%)	Gold Equiv. Metal (koz)	Gold Metal (koz)	Silver Metal (koz)	Copper Metal (kt)
Proved	1.76	2.83	2.57	10.71	0.07	163	146	722	2
Probable	50.31	1.49	1.38	5.47	0.04	2,462	2,237	8,841	21
Total	**52.07**	**1.57**	**1.42**	**5.64**	**0.04**	**2,625**	**2,383**	**9,563**	**23**

General Notes:

Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding

Mineral Resources are reported inclusive of Ore Reserves

1 Includes stockpiles

* Combined figure for V2 using 0.90g/t Au cut-off and A39 using 53 g/t Ag cut-off

Notes relevant to the gold equivalence calculation for silver and copper in the Mt Carlton Ore Reserve:

The calculation is based on commodity prices of A$1350/oz for gold, A$22.00/oz for silver and A$3.00/lb for copper

The calculation uses metallurgical recovery to concentrate of 89.0% for gold, 91.0% for silver and 91.0% for copper at V2 and 88.0% for silver and 92.0% for copper at A39 – based on recent plant performance

AuEq for Silver = ((Price Ag per oz x Ag Recovery)/(Price Au per oz)) x Ag Grade

AuEq for Copper = ((Price Cu per lb x 2204.623) x (Cu Recovery)) / ((Price Au per oz / 31.1034768) x (Cu Grade / 100)). Using a conversion factor of 1 Troy Ounce = 31.1034768 grams

All the elements included in the gold equivalent calculation (i.e. silver and copper) have been recovered and sold and there is a reasonable potential that this will continue to be the case.

Competent Person Notes refer to 1. Ian Patterson; 2. Guy Davies; 3. Tony Wallace

December 2013 Group Mineral Resource Statement

Gold			Measured			Indicated			Inferred			Total Resource			
Project	Type	Cut-Off	Tonnes (Mt)	Gold Grade (g/t)	Gold Metal (koz)	Tonnes (Mt)	Gold Grade (g/t)	Gold Metal (koz)	Tonnes (Mt)	Gold Grade (g/t)	Gold Metal (koz)	Tonnes (Mt)	Gold Grade (g/t)	Gold Metal (koz)	Competent Person
Cracow[1]	**Total**	**2.8**	**0.33**	**9.6**	**103**	**1.09**	**7.6**	**265**	**2.01**	**5.5**	**356**	**3.43**	**6.6**	**724**	**1**
Pajingo	Open-Pit	0.5	-	-	-	-	-	-	0.32	1.2	12	0.32	1.2	12	2
Pajingo[1]	Underground	2.5	0.11	13.1	46	2.68	6.6	564	1.74	5.4	301	4.51	6.3	911	2
Pajingo	**Total**		**0.11**	**13.1**	**46**	**2.68**	**6.6**	**564**	**2.06**	**4.7**	**313**	**4.84**	**5.8**	**923**	
Edna May[1]	Open-Pit	0.4	-	-	-	26.80	1.0	834	2.90	0.9	84	29.70	1.0	919	3
Edna May	Underground	3.0	-	-	-	-	-	-	1.30	5.4	226	1.30	5.4	226	3
Edna May	**Total**		**-**	**-**	**-**	**26.80**	**1.0**	**834**	**4.24**	**2.3**	**310**	**31.00**	**1.1**	**1,145**	
Mt Carlton	Open-Pit	0.35	-	-	-	10.4	2.4	807	-	-	-	10.40	2.4	807	4
Mt Carlton	Underground	2.5	-	-	-	-	-	-	0.77	4.7	115	0.77	4.7	115	4
Mt Carlton	Stockpile		0.19	1.6	9.69	-	-	-	-	-	-	0.19	1.6	10	
Mt Carlton	**Total**		**0.19**	**1.6**	**9.69**	**10.40**	**2.4**	**807**	**0.77**	**4.7**	**115**	**11.36**	**2.5**	**932**	
Mt Rawdon[1]	**Total**	**0.23**	**0.76**	**0.5**	**12**	**42.40**	**0.8**	**1,060**	**7.94**	**0.6**	**162**	**51.10**	**0.8**	**1,234**	**5**
Twin Hills[+]	Open-Pit	0.5	-	-	-	-	-	-	3.06	2.1	204	3.06	2.1	204	4
Twin Hills[+]	Underground	2.3	-	-	-	-	-	-	1.56	3.9	194	1.56	3.9	194	4
Twin Hills[+]	**Total**		**-**	**-**	**-**	**-**	**-**	**-**	**4.62**	**2.7**	**399**	**4.62**	**2.7**	**399**	
	Total		**1.19**	**4.5**	**171**	**83.36**	**1.3**	**3,530**	**21.60**	**2.4**	**1,655**	**106.35**	**1.6**	**5,356**	

Silver			Measured			Indicated			Inferred			Total Resource			
Project	Type	Cut-Off	Tonnes (Mt)	Silver Grade (g/t)	Silver Metal (koz)	Tonnes (Mt)	Silver Grade (g/t)	Silver Metal (koz)	Tonnes (Mt)	Silver Grade (g/t)	Silver Metal (koz)	Tonnes (Mt)	Silver Grade (g/t)	Silver Metal (koz)	
Mt Carlton	Open-Pit V2	0.35	-	-	-	10.40	23	7,690	-	-	-	10.40	23.0	7,690	4
Mt Carlton	Underground V2	2.5	-	-	-	-	-	-	0.77	15	371	0.77	15.0	371	4
Mt Carlton	Open-Pit A39	53 *	-	-	-	0.55	260	4,598	-	-	-	0.55	260	4,598	4
Mt Carlton	Stockpile		0.45	50	722	-	-	-	-	-	-	0.45	72	722	4
	Total		**0.45**	**50**	**722**	**10.95**	**35**	**12,288**	**0.77**	**15**	**371**	**12.3**	**34**	**13,381**	

Copper			Measured			Indicated			Inferred			Total Resource			
Project	Type	Cut-Off	Tonnes (Mt)	Copper Grade (%)	Copper Metal (kt)	Tonnes (Mt)	Copper Grade (%)	Copper Metal (kt)	Tonnes (Mt)	Copper Grade (%)	Copper Metal (kt)	Tonnes (Mt)	Copper Grade (%)	Copper Metal (kt)	
Mt Carlton	Open-Pit V2	0.35	-	-	-	10.40	0.3	28	-	-	-	10.40	0.3	28	4
Mt Carlton	Underground V2	2.5	-	-	-	-	-	-	0.77	0.3	3	0.77	0.3	3	4
Mt Carlton	Open-Pit A39	53 *	-	-	-	0.55	0.26	1	-	-	-	0.55	0.26	1	4
Mt Carlton	Stockpile		0.45	0.3	1.6	-	-	-	-	-	-	0.45	0.3	2	4
	Total		**0.45**	**0.3**	**1.6**	**10.95**	**0.3**	**29**	**0.77**	**0.3**	**3**	**12.3**	**0.28**	**34**	

Gold Equivalence	Tonnes (Mt)	Gold Equiv. Grade (g/t)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Grade (%)	Gold Equiv. Metal (koz)	Gold Metal (koz)	Silver Metal (koz)	Copper Metal (kt)
Measured	1.45	3.97	3.66	13.00	0.09	185	171	722	2
Indicated	83.91	1.40	1.31	4.55	0.04	3,843	3,530	12,288	29
Inferred	21.60	2.41	2.38	0.53	0.01	1,672	1,655	371	3
Total	**106.96**	**1.66**	**1.56**	**3.86**	**0.03**	**5,700**	**5,356**	**13,381**	**35**

General Notes:
Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding
Mineral Resources are reported inclusive of Ore Reserves
1 Includes stockpiles
* Ag cut-off for A39
+ Twin Hills has not changed as it is being reported as 2004 JORC Code

Notes relevant to the gold equivalence calculation for silver and copper in the Mt Carlton Mineral Resource:
The calculation is based on commodity prices of A$1350/oz for gold, A$22.00/oz for silver and A$3.00/lb for copper
The calculation uses metallurgical recovery to concentrate of 89.0% for gold, 91.0% for silver and 91.0% for copper at V2 and 88.0% for silver and 92.0% for copper at A39 – based on recent plant performance
AuEq for Silver = ((Price Ag per oz x Ag Recovery)/(Price Au per oz)) x Ag Grade
AuEq for Copper = ((Price Cu per lb x 2204.623) x (Cu Recovery)) / ((Price Au per oz / 31.1034768) x (Cu Grade / 100)). Using a conversion factor of 1 Troy Ounce = 31.1034768 grams
All the elements included in the gold equivalent calculation (i.e. silver and copper) have been recovered and sold and there is a reasonable potential that this will continue to be the case
Competent Person Notes refer to 1. Shane Pike; 2. Andrew Engelbrecht; 3. Greg Rawlinson; 4. Michael Andrew; and 5. Craig Bosel

Competent Persons Statement

The information in this statement that relates to the Mineral Resources or Ore Reserves listed in the table below is based on work compiled by the person whose name appears in the same row, who is employed on a full-time basis by Evolution Mining Limited and is a member of the institute named in that row. Each person named in the table below has sufficient experience which is relevant to the style of mineralisation and types of deposits under consideration and to the activity which he has undertaken to qualify as a Competent Person as defined in the JORC Code 2012. Noting however that the Twin Hills Mineral Resource was first disclosed under JORC Code 2004 requirements and has not been updated to JORC Code 2012 requirements Each person named in the table below consents to the inclusion in this report of the matters based on their information in the form and context in which it appears.

Activity	Competent Person	Institute
Cracow Mineral Resource	Shane Pike	Australasian Institute of Mining and Metallurgy
Cracow Ore Reserve	Ian Patterson	Australasian Institute of Mining and Metallurgy
Pajingo Mineral Resource	Andrew Engelbrecht	Australasian Institute of Mining and Metallurgy
Pajingo Ore Reserve	Ian Patterson	Australasian Institute of Mining and Metallurgy
Mt Rawdon Mineral Resource	Craig Bosel	Australasian Institute of Mining and Metallurgy
Mt Rawdon Ore Reserve	Tony Wallace	Australasian Institute of Mining and Metallurgy
Edna May Mineral Resource	Greg Rawlinson	Australasian Institute of Mining and Metallurgy
Edna May Ore Reserve	Guy Davies	Australasian Institute of Mining and Metallurgy
Mt Carlton Mineral Resource	Michael Andrew	Australasian Institute of Mining and Metallurgy
Mt Carlton Ore Reserve	Tony Wallace	Australasian Institute of Mining and Metallurgy
Twin Hills Mineral Resource	Michael Andrew	Australasian Institute of Mining and Metallurgy

MATERIAL INFORMATION SUMMARY

A Material Information Summary pursuant to ASX Listing Rules 5.8 and 5.9 is provided below for each of the Evolution mines together with commentary on changes between the December 2013 Mineral Resources and Ore Reserves and the previous position as at 31 December 2012. The Assessment and Reporting Criteria in accordance with JORC Code 2012 is presented in Appendix 1.

1.0 CRACOW

The December 2013 Cracow Mineral Resource estimate of 3.43Mt at 6.6g/t gold for 724koz ounces represents a decrease of 118koz net of mining depletion compared to the December 2012 estimate of 4.29Mt at 6.1g/t gold for 842koz. Changes are largely due to:

- Mining depletion during the period (-80koz gold)
- Increase in cut-off grade from 2.3g/t to 2.8g/t gold (-42koz)

Cracow Mineral Resources - December 2013												
Mineral Resource	Measured			Indicated			Inferred			Total Resource		
	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)
Royal	1.3	11.0	0.4	-	-	-	85.1	6.7	18.3	86.4	6.8	18.8
Crown	28.7	5.6	5.2	-	-	-	318.1	4.9	49.7	346.8	4.9	54.8
Klondyke	1.5	8.6	0.4	261.7	6.5	54.4	80.5	3.9	10.1	343.6	5.9	64.9
Sovereign	26.5	4.7	4.0	94.0	4.8	14.5	268.8	4.1	35.2	389.4	4.3	53.6
Kilkenny	57.8	9.8	18.2	244.3	7.1	55.7	561.5	5.2	94.6	863.7	6.1	168.5
Tipperary	12.8	4.6	1.9	207.4	8.1	54.2	132.0	5.8	24.4	352.2	7.1	80.5
Empire	35.0	16.8	18.9	147.4	11.6	54.9	143.8	8.5	39.3	326.2	10.8	113.2
Roses Pride	152.4	10.0	49.2	52.3	7.5	12.6	60.0	6.2	12.0	264.8	8.7	73.8
Phoenix	10.0	12.1	3.9	24.8	9.4	7.5	2.2	9.8	0.7	37.0	10.2	12.1
Coronation	-	-	-	-	-	-	308.8	6.1	61.0	308.8	6.1	61.0
Griffin	-	-	-	57.0	6.3	11.6	45.7	7.4	10.9	102.7	6.8	22.4
Stockpiles	6	5.1	1	-	-	-	-	-	-	6.1	5.1	1.0
Total	**332.0**	**9.6**	**103.2**	**1,089.0**	**7.6**	**265.3**	**2,006.5**	**5.5**	**356.0**	**3,428**	**6.6**	**724.5**

Notes:
Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding
Cracow Mineral Resources have been reported above an indicative cut-off grade of 2.8 g/t of gold
Mineral Resources are reported inclusive of Ore Reserves

The December 2013 Cracow Ore Reserve estimate of 1.36Mt at 5.9g/t gold for 260koz represents a minor decrease of 13koz net of mining depletion compared to the December 2012 estimate of 1.61Mt at 5.3g/t gold for 273koz. Changes are largely due to:

- Increase due to an upgrade of resources from Inferred to Indicated in Empire, Kilkenny, Phoenix Tipperary, and Roses Pride (+90koz)
- Mining depletion during the period (-91koz)

Cracow Ore Reserves - December 2013									
Ore Reserve	Proved			Probable			Total Reserve		
	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)
Kilkenny Upper	31.1	6.8	6.7	37.8	4.6	5.6	68.9	5.6	12.3
Kilkenny Lower	42.2	6.0	8.2	163.6	5.6	29.3	205.8	5.7	37.5
Tipperary	3.1	4.5	0.4	248.7	5.9	47.2	251.8	5.9	47.6
Phoenix	26.2	6.8	5.7	11.5	7.4	2.7	37.6	7.0	8.4
Griffin	-	-	-	11.7	5.2	2.0	11.7	5.2	2.0
Empire	50.9	11.4	18.6	286.5	5.8	53.3	337.4	6.6	71.9
Sovereign	-	-	-	17.0	5.3	2.9	17.0	5.3	2.9
Crown	1.7	8.3	0.5	0.1	1.4	0.0	1.9	7.9	0.5
Klondyke North	0.3	5.9	0.1	198.1	4.9	31.0	198.4	4.9	31.1
Royal	1.8	6.5	0.4	0.2	5.0	0.0	2.1	6.3	0.4
Roses Pride	189.9	6.8	41.8	24.9	2.8	2.2	214.8	6.4	44.0
Stockpile	6.0	5.1	1.0	-	-	-	6.0	5.1	1.0
Total	**353.1**	**7.3**	**83.4**	**1,000.1**	**5.5**	**176.2**	**1,354.3**	**5.9**	**259.6**

Notes:
Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding
Ore Reserves are reported above an indicative cut-off grade of 3.5 g/t gold

1.1 Cracow Mineral Resources

1.1.1 Geology and Geological Interpretation

The Cracow gold deposits are located within the Lower Permian Camboon Andesite, on the south-eastern flank of the Bowen Basin. The regional strike is north-northwest, dipping 20° west-southwest. The Camboon Andesite consists of andesitic and basaltic lava, with agglomerate, tuff and some inter-bedded trachytic volcanic rocks. Mineralisation is hosted in steeply dipping low sulphidation epithermal veins. These veins are discrete and as stock work, composed of quartz, carbonate and adularia with varying percentages of each mineral. Sulphide percentage in the veins are generally low (<3%) primarily composed of pyrite, with minor occurrences of hessite, sphalerite and galena. Alteration of the country rock can be extensive and zoned from the central veined structure.

1.1.2 Sampling and Sub-sampling

Numerous sample types were collected at Cracow and used in resource estimations. Predominately these were Diamond Drill Core, Rock Chip (hammer collection of development face samples) and Reverse Circulation (RC). A small number of samples from trenches/costeans were used in the Klondyke estimate.

Sample intervals for drill core and face samples were determined by visual logging of lithology type, veining style/intensity and alteration style/intensity to ensure a representative sample was taken. Sampling lengths ranged from a minimum of 0.4m to a maximum of 1.2m with sampling completed across the full width of mineralisation. RC samples were collected on 1m intervals.

Surface and underground drill core was halved with a core saw, with one half dispatched for analysis and the other half retained. All underground LTK60 was whole core sampled, with a small number of underground NQ holes whole core sampled during 2013. RC samples were collected with a cyclone and 7-1 split was taken at the drill rig using a riffle splitter.

Whole/half core samples were crushed in a jaw crusher to > 70% passing 2mm; half of this material was split with a riffle splitter for pulverising. No RC samples required crushing in the jaw crusher. Core and RC samples were pulverised for 10-14 minutes in a LM5 bowl with a target of 85% passing 75µm. From this material approximately 120g was scooped for further analysis and the remaining material re-bagged.

Sample preparation for rock chip face samples was conducted at the Cracow onsite laboratory. Samples were crushed in a Jaw Crusher to 100% passing 5mm; this material was then split with a riffle splitter and pulverised for 4 minutes in a LM2 bowl with a target of 85% passing 75µm. From this material 400g was collected with a scoop and sent for transport to another laboratory for assaying.

1.1.3 Sample Analysis Methods

The samples were analysed by 50g Fire Assay for gold with Atomic Absorption (AAS) finish. For silver an Aqua Regia digest with AAS finish was completed.

1.1.4 Drilling Techniques

A combination of drilling techniques was used across the Cracow lodes, including RC, Diamond HQ/NQ (triple tube and standard) and LTK60 drilling. Recording hole size or if the hole was drilled by diamond or RC techniques was sometimes missing in the older data. This uncertainty in the input data was considered when assigning resource categories to the blocks these particular holes informed.

1.1.5 Estimation Methodology

Geology (lithology and vein percent) along with gold grade were the principle controls for domaining which strongly influenced estimation. As the mineralisation at Cracow is hosted by discrete structures, mineralisation was domained and in some cases sub-domained into various lithology-grade domains, forming hard boundaries. These boundaries were used to constrain samples for estimation of blocks within these domains.

Ordinary Kriging (OK) was the preferred method of estimation used for Cracow Mineral Resources. In some cases other estimation techniques such as Inverse Distance were used. Variograms were generated using the composited drill-hole data, and search ellipses were orientated with the grade continuity identified by the variography.

1.1.6 Resource Classification

Resource categorisation was based on the confidence of the model, dependent but not limited to complexities relating to vein geometry and continuity, faulting, assay variability, data quality and associated QAQC. Drilling density also factored in the resource classification, with spacing varied across separate lodes and related to these complexities.

1.1.7 Cut-off Grade

The indicative Mineral Resource cut-off grade for Cracow is 2.8g/t, based on mining methods and associated mining and processing costs. For the 2013 Mineral Resource estimate, a gold price of A$1,350/oz was utilised.

1.1.8 Mining and Metallurgical Methods and Parameters and other modifying factors considered to date

See section 1.2.3 and 1.2.4 below.

1.2 Cracow Ore Reserve

1.2.1 Material Assumptions for Ore Reserves

The underground Ore Reserve is based on several assumptions which include:

- current minimum mining widths
- geotechnical similarities to current mining areas
- historical costs base for estimation of operating and capital costs
- historical metallurgical performance

Ore Reserves are not based on a fixed cut-off grade. Instead they are fully costed on historical unit cost data, modified for changing activity levels and realisation of recent cost saving initiatives.

1.2.2 Ore Reserve Classification

Classification of each stope panel is assessed on the proportion of tonnage in each resource classification. If the tonnage in the stope panel is greater than 50% Measured then it is classified as a Proved Ore Reserve. For stope panels greater than 50% Measured and Indicated it is classified as a Probable Ore Reserve. Stope panels that have greater than 50% of Inferred material are excluded from the Ore Reserve estimate.

This process leads to some Inferred resources being included in the Ore Reserve estimate as they will be mined as part of the Ore Reserve panels. This amount is less than 5% of the Ore Reserve estimate.

1.2.3 Mining Method

Mining of the Cracow ore bodies commenced in 2004. The mining method adopted is widely termed Modified Avoca. This is where stopes are extracted between levels based on geotechnical recommended parameters for stope lengths and heights, then backfilled with loose or consolidated fill before the next retreating stope is extracted. The method has been used extensively at Cracow throughout its ten year mine life. All deposits estimated in this report are amenable to this mining method.

1.2.4 Processing Method

The ore is to be processed through a traditional CIP/ CIL process plant at a current rate of approximately 550ktpa. The current and estimated future recoveries for gold are 94.0% and 80.0% for silver. An operating history of around ten years supports the metallurgical parameters used in the Ore Reserve estimation.

1.2.5 Cut-off Grade

Cut-off grades are not used to estimate Ore Reserves, they are more a generalisation of economic areas. There are numerous cut-off values dependent on cost structures applied. A fully costed stoping cut-off grade of 3.5g/t is representative of a mine cut-off grade.

All reserves are fully costed within an economic model and based on the proportion of operation and/or capital development required for ore extraction. Thus the cut-off grade varies dependent on these factors, and no one cut-off grade has been used for the Ore Reserves estimation.

1.2.6 Estimation Methodology

See section 1.1.5 above.

1.2.7 Material Modifying Factors

There are no concerning material modifying factors that need to be highlighted with the Ore Reserve. All regulatory leasing, approvals, licencing, agreements and current infrastructure are in place, which considers this estimation higher than that of a Feasibility Study.

2.0 PAJINGO

The December 2013 Pajingo Mineral Resource estimate of 4.84Mt at 5.8g/t gold for 923koz represents a decrease of 383koz net of mining depletion compared to the December 2012 estimate of 6.97Mt at 5.8g/t gold for 1,306koz. Changes are largely due to:

- Geological re-interpretation and optimised estimation parameters mainly at Jandam, Vera Underground, Janine and Cindy (-154koz) where more conservative modelling has reduced the search ellipse and increased the required drill density. This enables a better estimation of the resource where there is short-range variability in grade, orientation and thickness of veins.
- More stringent geotechnical parameters applied to mineralisation in close proximity to historic workings (-136koz)
- Mining depletion between reporting periods (-92koz)

Pajingo Mineral Resources - December 2013												
Mineral Resource	Measured			Indicated			Inferred			Total Resource		
	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)
Open-Pit												
Orchid	-	-	-	-	-	-	315.6	1.2	12.0	315.6	1.2	12.0
Underground												
Faith	4.2	10.8	1.5	32.7	9.2	9.7	181.3	4.9	28.8	218.2	5.7	39.9
Zed	10.6	5.6	1.9	381.0	6.6	81.1	358.5	4.6	52.9	750.1	5.6	135.9
Sonia East	19.6	13.0	8.2	35.3	10.9	12.3	108.4	6.4	22.3	163.3	8.2	42.9
Eva and Olivia	26.7	8.6	7.4	93.1	5.8	17.3	202.6	5.4	35.4	322.4	5.8	60.1
Veracity	-	-	-	37.9	7.3	8.8	131.0	5.6	23.4	168.9	5.9	32.3
Jandam	-	-	-	1,388.2	6.3	278.9	295.4	5.1	48.3	1,683.6	6.0	327.3
Vera	-	-	-	483.0	6.6	101.7	229.7	4.8	35.2	712.8	6.0	137.0
Sonia	34.2	23.7	26.0	88.7	9.6	27.2	172.2	8.3	45.7	295.2	10.4	99.0
Cindy	-	-	-	69.3	6.0	13.5	27.2	3.7	3.3	96.6	5.4	16.7
Janine	-	-	-	66.9	6.1	13.1	34.3	5.5	6.1	101.2	5.9	19.2
Stockpile	13.8	2.4	1.1	-	-	-	-	-	-	13.8	2.4	1.1
Total	**109.1**	**13.1**	**46.0**	**2,676.1**	**6.6**	**563.8**	**2,056.3**	**4.7**	**313.5**	**4,841.6**	**5.8**	**923.3**

Notes:
Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding
Pajingo Mineral Resources have been reported above an indicative cut-off grade of 2.5g/t gold for underground, 0.5g/t gold for open-pit and constrained to an A$1,350/oz pit design

The December 2013 Pajingo Ore Reserve estimate of 0.77Mt at 6.2g/t gold for 155koz represents a decrease of 174koz net of mining depletion compared to the December 2012 estimate of 1.62Mt at 6.3g/t gold for 329koz. Changes are largely due to:

- Mining depletion between reporting periods (-96koz)
- Additions from drilling in Zed and Sonia resulting in an upgrade of resource classification to Indicated category (+15koz)
- More conservative geological interpretation and estimation parameters mainly at Jandam, Vera Underground, Janine and Cindy (-77koz)
- Design changes due to further review of remnants of Vera South and Jandam areas (-15koz)

Pajingo Ore Reserves - December 2013									
Ore Reserve	Proved			Probable			Total Reserve		
	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (kt)	Grade Au (g/t)	Cont. Metal Au (koz)
Underground									
Cindy	2.3	3.4	0.3	23.2	5.4	4.0	25.5	5.2	4.3
Eva	30.1	5.1	4.9	43.7	4.6	6.4	73.7	4.8	11.3
Faith	1.2	5.9	0.2	5.1	7.1	1.2	6.4	6.8	1.4
Jandam	36.8	5.0	6.0	51.0	6.8	11.1	87.8	6.1	17.1
Sonia	41.6	11.3	15.2	58.0	6.3	11.7	99.6	8.4	26.9
Sonia East	26.3	8.3	7.0	30.1	8.9	8.6	56.3	8.6	15.6
Veracity	3.6	8.5	1.0	23.6	5.4	4.1	27.2	5.8	5.0
Vera South	11.9	9.2	3.5	60.7	6.0	11.7	72.5	6.5	15.2
Zed	10.5	4.3	1.5	300.2	5.7	55.4	310.7	5.7	56.8
Stockpile	13.8	2.3	1.0	-	-	-	13.1	2.3	1.1
Total	**178.1**	**7.1**	**40.5**	**595.5**	**6.0**	**114.2**	**773.5**	**6.2**	**154.7**

Notes:
Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding
Ore Reserves are reported above an indicative cut-off grade of 3.3 g/t gold

Given the current Mineral Resource inventory, the nature of the geology and the historic conversion of Mineral Resources to Ore Reserves at Pajingo, Evolution remains confident in the future of the operation and consistent with previous years has detailed schedules for mining over the next 5 years which is typical of the way the operation has run for many years.

2.1 Pajingo Mineral Resources

2.1.1 Geology and Geological Interpretation

Mineralisation is hosted in a series of structurally controlled sub-vertical, low-sulphidation epithermal quartz veins within an andesite host rock. Vein width ranges between 0.5m and 12m and extend up to 300m down dip and along an approximate strike – length of 2,100m for the various vein systems combined. Where multiple veins occur generally one vein contains the dominant proportion of metal. Veins have moderate to steep dips (60° - 90°) while width can vary rapidly along strike with down-dip width continuity being more consistent. Second order veins 10cm to 20cm wide commonly splay from the main structures and can extend 1m to 2m into the footwall or hanging wall.

2.1.2 Sampling and Sub-sampling

RC drilling was generally used to obtain 1m samples, each interval was logged by the geologist before determining intervals for analysis. A 2kg – 5kg sub-sample of the selected individual or composited sample intervals were obtained using a spear, and more recently a rig mounted cone splitter or riffle were used.

Surface diamond drill core was logged by the geologist who subsequently determines the required sample intervals. Most surface diamond drill core was sampled as half-core with a minimum sample interval of 0.2m and maximum sample interval of 1.5m.

Sampling of underground diamond drillholes followed the same protocol as surface drilling up to October 2013 after which whole core samples of nominal 1m length were submitted. Underground drillholes were assayed for gold and silver by fire assay.

Face sampling of underground development drives was routinely carried out as development advanced at 4m intervals, wall samples have also been taken where development has intersected mineralisation. Face and wall sampling involves a map being drawn and sample intervals determined bounded by lithology and alteration contacts (0.2m – 2m intervals). The geologist marks the contacts and/or sample intervals with paint and collects chips from within the interval directly into the sample bag.

2.1.3 Drilling Techniques

There has been a long history of drilling at Pajingo with methods and processes evolving with industry leading practices.

The majority of the current stated resource is defined by more recent data collected from underground, with surface drilling defining targets in the early stages of resource definition.

Underground diamond drillholes are typically 60mm in diameter; employing both wireline (NQ) and conventional drilling (LTK60) methods. 95% of underground drillholes are less than 300m in length. A small number of longer diamond holes have been drilled underground with a maximum length of 850m.

Underground face samples were taken as mining progressed in ore development drives, typically at 4m intervals. The drillhole represents a horizontal line of sampling (nominally 1.5m above the floor) across the exposed ore body and adjacent material.

2.1.4 Sample Analysis Methods

Core samples are submitted to the assaying laboratory where they were dried, coarse crushed to around 10mm and then pulverised to 85% passing 75µm. Subsamples were typically less than 3kg which allowed the total subsample to be prepared and pulverised. Quarter core field duplicates for surface diamond holes were based on at a ratio of 1:20 and showed a good correlation to primary assays. Quarter core and half core field duplicates were inserted with underground diamond holes at a ratio of 1:20 and show a good correlation to primary assays.

Core samples are submitted to the laboratory for preparation and fire assay reporting gold and silver values by fire assay and up to 50 additional elements by Inductively Coupled Plasma Optical Emission Spectrometry.

A small number of diamond drillholes have been sampled as whole core samples (after October 2013). The assaying laboratory dried, coarse crushed to ~10mm, split if >3kg and pulverised to 85% passing 75µm. Field duplicates were not submitted with whole core samples.

Prior to October 2013, face and wall samples were submitted for sample preparation and gold and silver analysis by fire assay. Samples were subsequently assayed by aqua regia for gold only.

2.1.5 Estimation Methodology

The gold grade estimation process was estimated by Ordinary Kriging (OK) and performed using Vulcan Maptek mining software. Inverse distance weighting interpolation was used to estimate grade into the blocks not informed by OK.

Waste (all material outside the mineralisation domain wireframes) was not estimated due to the highly clustered and limited of data. Therefore, waste was assigned to a below detection limit assay.

1m downhole composites using a minimum 0.3m length and aggregate merge method were generated for the drillhole dataset. Intervals with no assays were excluded from the composite data.

The following estimation parameters were optimised using Kriging Neighbourhood Analysis (KNA):

- Block size
- Number of samples
- Search range
- Block discretisation

The following estimation parameters were applied:

- Octant/quadrant search
- Two passes with on average 40%-60% of blocks estimated in the first pass for each domain
- Pass one with minimum of 8-12 samples and maximum of 16-24 samples
- Pass two with minimum 4-6 samples and maximum of 8-16 samples

- Search directions and ranges oriented to the variogram models of each mineralised domain
- Search range is applied 2/3 of the sill range
- Block discretisation of 10m x 5m x 10m (easting, northing, elevation)

Top cuts are applied to domains that have extreme values in the grade distribution. The top cut was defined by analysing log probability plots and the mean grade versus the Coefficient of Variance of the mineralised domains.

A formal peer review was performed by the internal Evolution resource group and the Pajingo Geology Manager by comparing the grade and tonnage of current resource estimates to the previous resource estimates. The current estimates take into account new data acquired from grade control and resource definition drilling together with up-dates in the geological interpretation of the deposits through the mapping of new development.

2.1.6 Resource Classification

Measured Mineral Resources are typically supported by drilling data which was mostly less than 20m x 20m spacing, and is additionally confirmed by underground development drives (face samples and geological backs mapping) and infill drilling between underground drives.

Indicated Mineral Resources are classified similar as Measured, but with less support from infill drilling and underground data. Typically drill spacing is less than 20m x 20m.

Inferred Mineral Resources is classified based on limited data support (no supported from underground data), less confident on the geological continuity, and typically drilling spacing is greater than 20m x 20m.

Other aspects that have been taken into account in defining the Mineral Resources classifications are:

- Data type and data quality (drill hole orientations, drill hole downhole surveys)
- Statistical performance of the estimate (i.e. slope regression, Kriging efficiency, number of samples/drill hole used)
- Geological underground backs mapping

2.1.7 Cut-off Grade

The Mineral Resource cut-off grade is estimated on historical cost data from July 2013 to March 2014, metallurgical assumptions on mill performance with allowance for gold royalties of 5%. The cut-off grade is reflective of a break-even stoping and milling cost.

2.1.8 Mining and Metallurgical methods, parameters and other modifying factors considered to date

See section 2.2.3 and 2.2.4 below.

2.2 Pajingo Ore Reserves

2.2.1 Material Assumptions for Ore Reserves

The underground Ore Reserve is based on several assumptions which include:

- current minimum mining widths
- geotechnical similarities to current mining areas
- historical costs base for estimation of operating and capital costs
- historical metallurgical performance

Ore Reserves are not based on a fixed cut-off grade. Instead they are fully costed on historical unit cost data, modified for changing activity levels and realisation of recent cost saving initiatives.

2.2.2 Ore Reserve Classification

Classification of each stope panel is assessed on the proportion of tonnage in each resource classification. If the tonnage in the stope panel is greater than 50% Measured then it is classified as a Proved Ore Reserve. For stope panels greater than 50% Measured and Indicated it is classified as a Probable Ore Reserve. Stope panels that have greater than 50% of Inferred material are excluded from the Ore Reserve estimate.

This process leads to some Inferred Resources being included in the Ore Reserve estimate as they will be mined as part of the Ore Reserve panels. This amount is less than 5% of the Ore Reserve estimate.

2.2.3 Mining Method

The mining method adopted is widely termed Modified Avoca whereby stopes are extracted between levels based on geotechnical recommended parameters for stope lengths and heights, then backfilled with loose or consolidated fill before the next retreating stope is extracted. The method has been used extensively at Pajingo throughout its 20 year mine life.

Mineable panels have been created based on typical level intervals for the Modified Avoca mining method currently used at the operation. The resource models have been divided into stope panels 20m long at level interval height (15-20m). No geotechnical evaluation of the reserves has been undertaken, however geotechnical parameters are based on current stoping practices and not expected to diverge greatly from these assumptions.

All stopes panels have a minimum stoping width of 1.8m which, dependent on ore width, is considered as planned dilution. An additional 25% external dilution at a grade of 0.0g/t has been applied to all stopes. This allows for stope overbreak, floor and rill dilution in stope ore extraction. Mining recovery of 95% is estimated for stope panels based on current experience at the underground operation. The current underground infrastructure is suitable to support the mining method with extensions to existing capital development.

2.2.4 Processing Method

Ore is processed through a traditional carbon-in-leach / carbon-in-pulp process plant at a current rate of around 450ktpa. The current and estimated future recoveries are 95%. An operating history of over 20 years supports the metallurgical parameters used in the Ore Reserve estimation. Metallurgical characterisation was conducted in late 2012 on representative samples of current and extensional ore bodies, which confirmed the current metallurgical parameters.

An operating history of over 20 years supports the metallurgical parameters used in the Ore Reserve estimation. Metallurgical characterisation was conducted in late 2012 on representative samples of current and extensional ore bodies, confirmed the current metallurgical assumptions.

2.2.5 Cut-off Grade

Cut-off grades are not used to estimate Ore Reserves, they are more a generalisation of economic areas. There are numerous cut-off values dependent on cost structures applied. A fully costed stoping cut-off grade of 3.3g/t is representative of a mine cut-off grade.

All reserves are fully costed within an economic model and based on the proportion of operation and/or capital development required for ore extraction. Thus the cut-off grade varies dependent on these factors, and no one cut-off grade has been used for the Ore Reserves estimation.

2.2.6 Estimation Methodology

See section 2.1.5 above.

2.2.7 Material Modifying Factors

There are no concerning material modifying factors that need to be highlighted with the Ore Reserve. All regulatory leasing, approvals, licencing, agreements and current infrastructure are in place, which considers this estimation higher than that of a feasibility study.

3.0 EDNA MAY

The December 2013 Edna May Mineral Resource estimate of 31.0Mt at 1.1g/t gold for 1,145koz represents a decrease of 498koz net of mining depletion compared to the December 2012 estimate of 47.0Mt at 1.1g/t gold for 1,643koz. Changes are largely due to:

- Decrease due to reporting Mineral Resources within a pit optimisation shell based on a long term gold price assumption of A$1,800 per ounce (-174koz)
- Decrease due to reporting Mineral Resources within underground stope shapes constructed according to a long term gold price assumption of A$1,800 per ounce (-48koz)
- Geological re-interpretation and re-estimation of Greenfinch resource using defined wireframes (-114koz) and exclusion of mineralisation outside the Edna May Gneiss geological domain (-63koz) pending additional drilling and geological modelling
- Mining depletion during the period (-90koz)
- Stockpile changes due to opening and closing stocks within reporting period (-10koz)

Planned work at Edna May will focus on the potential to improve the geological understanding of the Greenfinch deposit and other areas of the Edna May deposit to determine whether the mineralisation excluded from the current Mineral Resource estimate can be included in the next revision.

Edna May Mineral Resources - December 2013												
Mineral Resource	Measured			Indicated			Inferred			Total Resource		
	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)
Open-Pit												
Edna May	-	-	-	23.35	1.0	758	2.81	0.9	81	26.16	1.0	840
Greenfinch	-	-	-	1.47	0.9	40	0.13	0.8	3	1.60	0.8	44
Underground												
Edna May	-	-	-	-	-	-	1.30	5.4	226	1.30	5.4	226
Stockpile	-	-	-	1.96	0.6	36	-	-	-	1.96	0.6	36
Total	**-**	**-**	**-**	**26.78**	**1.0**	**834**	**4.24**	**2.3**	**310**	**31.03**	**1.1**	**1,145**

Notes:
Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding
Edna May and Greenfinch Mineral Resources have been reported above a cut-off grade of 0.4g/t gold and Edna May underground reported above 3.0g/t gold
Edna May open-pit was reported within an optimised shell based on a $1,800/oz gold price
Greenfinch was reported within an optimised shell based on a $1,800/oz gold price
Edna May underground deposit is reported within a nominal optimised envelope
Mineral Resources are reported inclusive of Ore Reserves

The December 2013 Edna May Ore Reserve estimate of 11.3Mt at 1.1g/t gold for 402koz represents a decrease of 307koz net of mining depletion compared to the December 2012 estimate of 22.5Mt at 1.0g/t gold for 709koz. Changes are largely due to:

- Decrease due to geological re-interpretation and re-estimation of Greenfinch resource (-86koz) and exclusion of mineralisation outside the Edna May Gneiss geological domain (-34koz) pending additional drilling and geological modelling
- Mining depletion during the period (-76koz)
- Mining design adjustment due to the use of a lower gold price (A$1,350/oz) (-86koz)
- Decrease due to change in cut-off grade (-15koz)
- Stockpile changes due to opening and closing stocks within reporting period (-10koz)

Edna May Ore Reserves - December 2013									
Ore Reserve	Proved			Probable			Total Reserve		
	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)
Open-Pit									
Edna May	-	-	-	9.35	1.2	366	9.35	1.2	366
Stockpile	-	-	-	2.00	0.6	36	2.00	0.6	36
Total	**-**	**-**	**-**	**11.35**	**1.1**	**402**	**11.35**	**1.1**	**402**

Notes:
Data is reported to significant figures and differences may occur due to rounding
Ore Reserves are reported above a 0.5g/t gold cut-off

3.1 Edna May Mineral Resources

3.1.1 Geology and Geological Interpretation

The Edna May Gold Mine is situated to the west-northwest end of the Westonia Greenstone Belt, within the Archaean Southern Cross Province of Western Australia. The deposit is hosted within the Edna May Gneiss (one of several host units locally), between an ultra-mafic hanging wall and meta-basalt footwall. The three main Gneiss units in the region are the Edna May, Golden Point and Greenfinch Gneiss's. The strata dips approximately 50 degrees to the north. The Edna May deposit comprises high-grade quartz reefs with halo mineralisation hosted in the gneiss unit. The majority of the gold occurs within quartz veins, with lesser amounts in alteration halos. Two types of veins are noted but the more important type, in terms of historic gold production, comprises of a series of stacked veins that form splays from the footwall shear zone.

3.1.2 Sampling and Sub-Sampling

RC and diamond core was sampled. Diamond core recoveries have been logged recorded with an average of approximately 95%. Diamond core is reconstituted into continuous runs for orientation marking and recovery estimations. Core loss (if any) is recorded. RC drill sample recoveries were not recorded. Historically RC samples were collected at 1m intervals in individually marked calico bags through a three tier riffle or cone splitter.

Geological logging has been carried out for each drill hole. This includes lithology grain size, mineralisation, alteration, sulphides and oxidation. Core was cut in half and sampled on intervals between 0.2m and 1.2m.

RC drilling was completed over several generations. Sampling consisted of three tier riffle splitters or cone splitters. The sample preparation technique for RC and diamond is considered to be of standard practice within the industry and deemed appropriate.

Pre-Catalpa Resources data was utilised on the basis of existing documented historic quality control practices. Later stage drilling follows Catalpa Resources internal quality control practice which includes a review of laboratory supplied blanks and standards as well as Catalpa supplied blanks and standards.

3.1.3 Sample Analysis Methods

Sample analysis has been carried out at various commercial laboratories over the history of the deposit. RC and diamond samples were either sampled using either screen fire assay, fire assay with a 50g charge, fire assay with a 30g charge or aqua regia techniques.

3.1.4 Drilling Techniques

Edna May has an extensive history of generations of drilling over the life of the region. The Edna May resource is estimated from the data of 1,372 RC and 335 diamond holes since the mid 1980's with the much of drilling completed within the last ten years.

3.1.5 Estimation Methodology

Gold mineralisation at Edna May is generally bimodal in nature due to higher grade reefs and halo material and lower grade disseminated mineralisation in the surrounding country rock. The ore deposit has been divided into four domains based on statistical relationships on grade populations. Samples size is not consistent between diamond core and RC sampling therefore all samples are composited to 2.5m. Composited drill samples were flagged as within or outside the domain wire frames. The estimation technique known as Multiple Indicator Kriging (MIK) has been used for Edna May Open cut, while Ordinary Kriging was used for Greenfinch and Edna May underground resources.

3.1.6 Resource Classification

The resource estimate within each panel have been classified according to distribution of sampling in the kriging neighbourhood. The result has been reviewed qualitatively to ensure it appears realistic and has been downgraded if it appeared optimistic. Indicatively, areas with a drill density of 25m by 25m spacing have been classified as indicated.

3.1.7 Cut-off Grade

The cut-off grade used to report the Mineral Resources at Edna May and Greenfinch is 0.4g/t gold. The cut-off grade used for reporting the Edna May Underground Mineral Resource is 3.0g/t gold.

3.1.8 Mining and Metallurgical methods, parameters and other modifying factors considered to date

See sections 2.2.3 and 2.2.4 below.

3.2 Edna May Ore Reserves

3.2.1 Material Assumptions for Ore Reserves

The Edna May open pit Ore Reserve estimate is formulated by applying the Whittle Lerchs-Grossman algorithms to the Mineral Resource model using current and forecasted cost structures, revenue, recovery and geotechnical parameters. A detailed pit design derived from the selected optimum shell limits is used to calculate the Ore Reserve estimate and mining depletion as at 31 December 2013 is subtracted. The open pit Ore Reserves are defined using a block grade cut-off approach and are inclusive of low grade stockpiles. The current strategy at Edna May involves open pit mining of the main pit in three stages by conventional drill and blast, excavator and truck activities.

3.2.2 Ore Reserve Classification

All of the Ore Reserves are currently derived from Indicated Resources, this includes both in-situ material and existing stockpiles.

3.2.3 Mining Method

Current mining activities at Edna May are undertaken via a conventional drill and blast, truck and excavator open pit operation with 10m high blasting benches mined in three flitches of 3m, 3.5m and 3.5m respectively. The Edna May pit will be developed in three stages, the initial stage 1 pit and a southern and northern cutback. As the block modelling estimation methodology takes into account mining dilution and recovery loss these factors have not been applied to the Edna May Ore Reserve estimate. Waste material is classified as material less than the marginal cut-off grade (0.5g/t au) and will either be transported to the raising of the integrated waste landform (IWL) for tailings disposal or a typical waste rock dump. Ore is classified as material greater than the marginal cut-off grade (0.5g/t au) and depending on the scheduled stockpiling strategy will be taken to the Run of Mine (ROM) pad for immediate processing or low grade stockpile for future processing.

The current operations demonstrate the appropriateness of this mining method as the basis of the Ore Reserve estimate.

3.2.4 Processing method

The Edna May ore is processed through a conventional crush, grind, carbon in leach (CIL) circuit at a rate of 2.9Mtpa. Gold doré is produced at the final stage of the process.

A metallurgical recovery rate of 93.0% has been applied in the Ore Reserve estimate as although historic gold recoveries are 92.0%, plant modifications to lift gold recovery are in progress.

No assumptions or allowances have been made for deleterious elements as these elements are not anticipated to impact the process or value of the ore.

3.2.5 Cut-off Grade

The marginal cut-off grade used to report the Ore Reserves is derived from the cost of processing ore (including site general and administration costs), additional incremental ore mining costs, metallurgical recoveries, royalties and gold price. A cut-off grade of 0.5g/t gold has been used for the Ore Reserve estimate.

3.2.6 Estimation Methodology

See section 3.1.5 above.

3.2.7 Material Modifying Factors

There are no concerning material modifying factors that need to be highlighted with the Ore Reserve. All regulatory leasing, approvals, licensing, agreements and current infrastructure are in place, which considers this estimation higher than that of a feasibility study.

4.0 MT CARLTON

The Mt Carlton Mineral Resource consists of the V2 gold-silver-copper deposit and the A39 silver-copper deposit and stockpiled material.

The December 2013 Mt Carlton Mineral Resource estimate for V2 of 11.4Mt at 2.5 g/t gold, 22g/t silver and 0.27% copper for 932koz gold, 8.18Moz silver, and 31kt copper (1.19Moz gold equivalent) represents a decrease of 418koz gold, 6.9Moz silver and 28.7kt copper (0.65Moz gold equivalent) net of mining depletion compared to the December 2012 estimate of 25.2Mt at 1.7g/t gold, 19g/t silver and 0.24% copper for 1,350koz gold, 15.1Moz silver and 59.7kt copper.

Changes to the Mineral Resource estimate for the V2 deposit are largely due to:

- The new resource is constrained by an optimised pit shell based on commodity prices of A$1,800/oz gold, A$28/oz silver and A$3.00/lb copper. Mineralised material outside of the optimised shell was assessed with a stope optimisation process to identify potential underground resources, for material above a 2.5g/t Au cut-off. The constraint of the resources within nominal economic envelopes resulted in a reduction of 343koz gold, 6.4Moz silver and 26.6kt copper from the previously reported resource (678koz gold equivalent)
- Mining depletion at V2 during the period (-45koz Au, -740koz Ag, and 2,100t Cu for -65koz gold equivalent)

Open pit mining of the V2 deposit has revealed geologic complexity beyond the resolution of the drilling undertaken for the original resource estimate. The positive reconciliation, coupled with unexpected structurally controlled zones of high-grade gold mineralisation has necessitated an infill drill programme to better define smaller-scale but important controls. This coupled with other work on alteration and sulphide mineral association will in due course result in a new model that in addition to improving the geology will help inform a blending strategy.

The December 2013 Mt Carlton Mineral Resource estimate for A39 of 0.9Mt at 197g/t silver and 0.33% copper for 5.20Moz silver and 2kt copper represents a decrease of 10.9Moz silver and 1.4kt copper (163koz gold equivalent) net of mining depletion compared to the December 2012 estimate of 2.9Mt at 170g/t silver and 0.12% copper for 16.1Moz silver and 3.4kt of copper. Further studies are planned to assess the potential for extending the A39 resource as an open-cut or underground operation.

Changes to the Mineral Resource estimate for the A39 deposit are largely due to:

- The new resource is constrained by an optimised pit shell based on commodity prices of A$28/oz silver and A$3.00/lb copper. The constraint of the resources within nominal economic envelopes resulted in a reduction of 2.7Moz silver, 1.5kt copper (45koz gold equivalent) from the previously reported resource
- Mining depletion at A39 during the period 8.2Moz Ag, 0.6kt Cu (120koz gold equivalent)

Gold - Mt Carlton Mineral Resources - December 2013												
Mineral Resource	**Measured**			**Indicated**			**Inferred**			**Total Resource**		
	Tonnes (Mt)	**Grade Au (g/t)**	**Cont. Metal Au (koz)**	**Tonnes (Mt)**	**Grade Au (g/t)**	**Cont. Metal Au (koz)**	**Tonnes (Mt)**	**Grade Au (g/t)**	**Cont. Metal Au (koz)**	**Tonnes (Mt)**	**Grade Au (g/t)**	**Cont. Metal Au (koz)**
A39 Open-cut	-	-	-	-	-	-	-	-	-	-	-	-
V2 Open-cut	-	-	-	10.4	2.4	807	0.77	4.7	115	11.17	2.6	922
V2 Stockpile	0.19	1.6	9.69	-	-	-	-	-	-	0.19	1.6	10
Total	**0.19**	**1.6**	**9.69**	**10.4**	**2.4**	**807**	**0.77**	**4.7**	**115**	**11.36**	**2.5**	**932**

Silver - Mt Carlton Mineral Resources - December 2013												
Mineral Resource	**Measured**			**Indicated**			**Inferred**			**Total Resource**		
	Tonnes (Mt)	**Grade Ag (g/t)**	**Cont. Metal Ag (koz)**	**Tonnes (Mt)**	**Grade Ag (g/t)**	**Cont. Metal Ag (koz)**	**Tonnes (Mt)**	**Grade Ag (g/t)**	**Cont. Metal Ag (koz)**	**Tonnes (Mt)**	**Grade Ag (g/t)**	**Cont. Metal Ag (koz)**
A39 Open-cut	-	-	-	0.55	260	4,598	-	-	-	0.6	260	4,598
A39 Stockpile	0.26	72	606	-	-	-	-	-	-	0.3	72	606
V2 Open-cut	-	-	-	10.40	23	7,690	-	-	-	10.4	23	7,690
V2 Underground	-	-	-	-	-	-	0.77	15	371	0.8	15	371
V2 Stockpile	0.19	19	116	-	-	-	-	-		0.2	19	116
Total	**0.45**	**50**	**722**	**10.95**	**35**	**12,288**	**0.77**	**15**	**371**	**12.3**	**34**	**13,381**

Copper - Mt Carlton Mineral Resources - December 2013												
Mineral Resource	**Measured**			**Indicated**			**Inferred**			**Total Resource**		
	Tonnes (Mt)	**Grade Cu (%)**	**Cont. Metal Cu (kt)**	**Tonnes (Mt)**	**Grade Cu (%)**	**Cont. Metal Cu (kt)**	**Tonnes (Mt)**	**Grade Cu (%)**	**Cont. Metal Cu (kt)**	**Tonnes (Mt)**	**Grade Cu (%)**	**Cont. Metal Cu (kt)**
A39 Open-cut	-	-	-	0.6	0.26	1	-	-	-	0.6	0.26	1
A39 Stockpile	0.26	0.5	1.28	-	-	-	-	-	-	0.3	0.49	1
V2 Open-cut	-	-	-	10.4	0.27	28	-	-	-	10.4	0.27	28
V2 Underground	-	-	-	-	-	-	0.77	0.33	3	0.8	0.33	3
V2 Stockpile	0.19	0.16	0.3							0.2	0.16	0.3
Total	**0.45**	**0.3**	**1.58**	**11.0**	**0.27**	**29**	**0.77**	**0.33**	**3**	**12.3**	**0.28**	**33.3**

Gold Equivalent - Mt Carlton Mineral Resources - December 2013									
Mineral Resource	**Tonnes (Mt)**	**Grade AuEq (g/t)**	**Grade Au (g/t)**	**Grade Ag (g/t)**	**Grade Cu (g/t)**	**Cont. Metal AuEq (koz)**	**Au Metal (koz)**	**Ag Metal (koz)**	**Cu Metal (kt)**
Measured	0.5	1.67	0.67	41.88	0.28	24	10	722	1
Indicated	11.0	2.99	2.29	34.90	0.27	1,119	807	12,288	29
Inferred	0.8	5.33	4.65	15.00	0.33	132	115	371	3
Total	**12.3**	**3.26**	**2.38**	**33.90**	**0.27**	**1,278**	**932**	**13,381**	**33.3**

Notes:

Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding

Mt Carlton V2 deposit Mineral Resources have been reported above a cut-off grade of 0.35g/t gold and A39 deposit reported above 53g/t silver. The open cut resources have been reported within an optimised shell based on an A$1,800/oz gold price, A$28/oz silver price and A$3.00lb copper price. The V2 underground resource has been constrained within an optimised shell based on a 2.5 g/t gold cut-off

Mineral Resources are reported inclusive of Ore Reserves

Notes relevant to the gold equivalence calculation for silver and copper in the Mt Carlton Mineral Resource:

The calculation is based on commodity prices of A$1,350/oz for gold, A$22.00/oz for silver and A$3.00/lb for copper

The calculation uses metallurgical recovery to concentrate of 89.0% for gold, 91.0% for silver and 91.0% for copper at V2 and 88.0% for silver and 92.0% for copper at A39 – based on recent plant performance

AuEq for Silver = ((Price Ag per oz x Ag Recovery)/(Price Au per oz)) x Ag Grade

AuEq for Copper = ((Price Cu per lb x 2204.623) x (Cu Recovery)) / ((Price Au per oz / 31.1034768) x (Cu Grade / 100)). Using a conversion factor of 1 Troy Ounce = 31.1034768 grams

All the elements included in the gold equivalent calculation (i.e. silver and copper) have been recovered and sold and there is a reasonable potential that this will continue to be the case.

The December 2013 Mt Carlton Ore Reserve estimate for V2 of 7.30Mt at 3.0g/t gold, 22g/t silver and 0.28% copper for 705koz gold, 5.04Moz silver and 20.3kt copper (870koz gold equivalent) represents a decrease of 107koz gold, 1.42Moz silver and 4.5kt copper (148koz gold equivalent) net of mining depletion compared to the December 2012 estimate of 9.13Mt at 2.8g/t gold, 22g/t silver and 0.27% copper for 812koz gold, 6.46Moz silver and 24.8kt copper.

Changes to the Ore Reserve estimate for the V2 deposit are largely due to:

- Decrease due to geological re-interpretation and optimisation of estimation parameters (-62koz Au, -337koz Ag, and -1,275t Cu (-73koz gold equivalent)
- Mining depletion during the period (-41koz Au, -546koz Ag, and -1,714t Cu (-57koz gold equivalent)
- Decrease due to grade cut-off increase from 0.69g/t Au to 0.90g/t Au (-22koz Au, -720koz Ag and -1,944t Cu (-41 koz gold equivalent)

The December 2013 Mt Carlton Ore Reserve estimate for A39 of 0.67Mt at 209g/t silver and 0.38% copper for 4.53Moz silver and 2.6kt copper represents a decrease of 4.32Moz silver and increase of 0.1kt copper (62koz gold equivalent decrease) net of mining depletion compared to the December 2012 estimate of 1.04Mt at 265g/t silver and 0.24% copper for 8.85Moz silver and 2.5kt copper.

Changes to the Ore Reserve estimate for the A39 deposit are largely due to:

- Mining depletion during the period (-4,310koz Ag and -995t Cu (-66koz gold equivalent)
- Pit design change (-623koz Ag,-184t Cu (-10koz gold equivalent)

Gold - Mt Carlton Ore Reserves - December 2013									
Ore Reserve	Proved			Probable			Total Reserve		
	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)
A39 Open-cut	-	-	-	-	-	-	-	-	-
V2 Open-cut	-	-	-	7.11	3.0	695	7.11	3.0	695
V2 Stockpile	0.19	1.6	10	-	-	-	0.2	1.6	10
Total	**0.19**	**1.6**	**10**	**7.11**	**3.0**	**695**	**7.30**	**3.0**	**705**

Silver - Mt Carlton Ore Reserves - December 2013									
Ore Reserve	Proved			Probable			Total Reserve		
	Tonnes (Mt)	Grade Ag (g/t)	Cont. Metal Ag (koz)	Tonnes (Mt)	Grade Ag (g/t)	Cont. Metal Ag (koz)	Tonnes (Mt)	Grade Ag (g/t)	Cont. Metal Ag (koz)
A39 Open-cut	-	-	-	0.41	295	3,920	0.41	296	3,920
V2 Open-cut	-	-	-	7.11	22	4,921	7.11	22	4,921
A39 Stockpile	0.26	72	606	-	-	-	0.26	72	606
V2 Stockpile	0.19	19	116	-	-	-	0.19	19	116
Total	**0.45**	**50**	**722**	**7.52**	**37**	**8,841**	**7.97**	**38**	**9,563**

Copper - Mt Carlton Ore Reserves - December 2013									
Ore Reserve	Proved			Probable			Total Reserve		
	Tonnes (Mt)	Grade Cu (%)	Cont. Metal Cu (kt)	Tonnes (Mt)	Grade Cu (%)	Cont. Metal Cu (kt)	Tonnes (Mt)	Grade Cu (%)	Cont. Metal Cu (kt)
A39 Open-cut	-	-	-	0.41	0.31	1.3	0.41	0.31	1.3
V2 Open-cut	-	-	-	7.11	0.28	20.0	7.11	0.28	20.0
A39 Stockpile	0.26	0.49	1.3	-	-	-	0.26	0.49	1.3
V2Stockpile	0.19	0.16	0.3	-	-	-	0.19	0.16	0.3
Total	**0.45**	**0.35**	**1.58**	**7.52**	**0.28**	**21.3**	**7.97**	**0.33**	**23.0**

Gold Equivalent - Mt Carlton Ore Reserves - December 2013									
Ore Reserve	Tonnes (Mt)	Gold Equiv. Grade (g/t)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Grade (g/t)	Gold Equiv. Metal (koz)	Gold Metal (koz)	Silver Metal (koz)	Copper Metal (kt)
Proved	0.45	1.87	1.6	50	0.28	27	10	722	1.3
Probable	7.52	3.81	3.0	37	0.28	920	695	8,841	21.3
Total	**7.97**	**3.70**	**3.0**	**38**	**0.28**	**947**	**705**	**9,563**	**23.0**

Notes:

Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding
V2 Ore Reserves are reported above a 0.90g/t gold cut-off and A39 Ore Reserves reported above a 53g/t silver cut-off

Notes relevant to the gold equivalence calculation for silver and copper in the Mt Carlton Ore Reserve:

The calculation is based on commodity prices of A$1,350/oz for gold, A$22.00/oz for silver and A$3.00/lb for copper
The calculation uses metallurgical recovery to concentrate of 89.0% for gold, 91.0% for silver and 91.0% for copper at V2 and 88.0% for silver and 92.0% for copper at A39 – based on recent plant performance
AuEq for Silver = ((Price Ag per oz x Ag Recovery)/(Price Au per oz)) x Ag Grade
AuEq for Copper = ((Price Cu per lb x 2204.623) x (Cu Recovery)) / ((Price Au per oz / 31.1034768) x (Cu Grade / 100)). Using a conversion factor of 1 Troy Ounce = 31.1034768 grams
All the elements included in the gold equivalent calculation (i.e. silver and copper) have been recovered and sold and there is a reasonable potential that this will continue to be the case.

4.1 Mt Carlton Mineral Resources

4.1.1 Geology and Geological Interpretation

The Mt. Carlton project covers the northern margin of the Permian Bowen Basin, in particular the basal Lizzie Creek Volcanics with minor Back Creek Group sediments.

Mineralisation is hosted in the basal sequence of felsic to silicic volcanics un-conformably overlying the Lower Carboniferous Glen Alpine Adamellite. Mineralisation is hosted exclusively within Rhyodactite volcanics. The Rhyodacites have been cross cut by numerous steeply dipping basic dykes. Gold, silver, and copper mineralisation occurs in strata-bound silicified layers, stock-works, breccia zones, and in banded epithermal veins with mineralisation primarily as enargite, polybasite and some native gold.

Two distinct areas of mineralisation occur. To the west is the silver rich A39 area in a vertical, east striking fault containing high grade epithermal silver veins. The second distinct area of mineralisation at Silver Hill is a gold, copper-silver sulphides zone, known as V2. Mineralisation occurs as matrix in-fill to a north dipping breccia or fracture zone at the intersection of north north-east and north north-west trending enargite-pyrite veins.

Mineralisation lies in the fresh rock under 20-25m of oxidised and weathered cover.

4.1.2 Sampling and Sub-sampling

RC samples were collected using cone splitter at 1m intervals. All samples were collected dry. Field duplicates were collected in the same manner as original samples at a frequency of 1 in 20. RC and diamond core were logged for lithology, alteration, texture, weathering and mineralisation. Texture and structure data were recorded for core only. Core was routinely photographed after logging. Core was cut using a core saw and sampled at nominal one meter intervals from the same side in the tray at all times.

Samples were also collected using geological controls at preferential intervals. Core was cut in half through marked orientation lines or on core axis. Quarter core was taken where check samples were required whiles whole core was taken for geotechnical testing. Geotechnical logging was undertaken for oriented core, data collected included; core recovery, RQD, weathering, alteration, estimated rock strength, joint spacing, joint condition, lithological description/units, number of defects, defect type, roughness, infill and infill thickness.

4.1.3 Sample Analysis Methods

Core and RC samples were dried and crushed at ALS Chemex, Townsville and SGS Laboratories in Townsville using industry best practice. Samples were pulverized at nominal 85% passing 75microns for assaying.

Certified Reference Materials inserted into sample stream covered 5% of sample volume. These included standards, blanks and field duplicates. Initial assays were conducted on four metre composites of initial one metre samples taken during drilling. Significantly mineralised intervals are subsequently re-assayed using 1 metre field re-splits from RC cuttings retained on site.

Analytical procedures used by both ALS (Aqua Regia/ICPAES, MEOG46) and SGS (ICP24R) for base metals were partial digestion methods using 2-Acid. Gold was analysed using 50 fire assay charge by both laboratories. ALS - AUAA-26, SGS - FAL505. Certified Reference Materials (standards, blanks, split duplicates) formed part of the routine internal laboratory QAQC procedure. Accuracy and precision of QAQC data was monitored using control charts. Checks on assay accuracy and umpire analysis were conducted at ALS and SGS. Results indicate good reproducibility and contamination was contained.

4.1.4 Drilling Techniques

RC and diamond drilling (DD – HQ diameter) methods were used to sample the both V2 and A39 Resource areas. Data for the current estimates were collected from 2006 to 2011. Holes were drilled on 50m centres angled steep to grid south to vertical across most part of the deposit. Areas of significant mineralisation were in-filled to 25m x 25m spacing.

4.1.5 Estimation Methodology

At V2 an “E-type” estimate was used for the current model with a block dimension of 10mX10mX5m. At A39 a combination of MIK, Ordinary Kriging (OK) and Conditional Simulation (CS) models have been used to produce the final block grade estimates in the current study. In addition to the estimation of economic and secondary deleterious metals the new Mt Carlton resource model incorporates details of weathering and major lithology (including barren dykes) for both mineralised and waste blocks.

Five Domains (two at A39, two at V2 and one domain for weak mineralisation) were used to constrain the estimate. The estimate was defined using 609 drill holes predominantly RC largely drilled on a 25m x 25m grid. These were composited into 44,585 2m composites for gold, silver, copper and arsenic estimation, 47,277 composites were used to estimate zinc, bismuth, antimony and sulphur (minor elements). Iron was under sampled compared to the other metals, having 45,834 composites.

A further 176 vertical grade control holes spaced at 10m x 10m produced 7,674 one metre down hole composites from A39 deposit. A top-cut of 90g/t gold and 7,000g/t silver was applied.

A39 and minor elements were estimated using a combination of OK for widely spaced drill data and Sequential Gaussian Simulation for more closely spaced grade control drilled areas of the deposit into 10m x10m x 2.5m blocks.

4.1.6 Resource Classification

Blocks in the resource model have been allocated a confidence category based on the number and location of samples used to estimate the grade of each block in the MIK models. The approach is based on the principle that larger numbers of samples, which are more evenly distributed throughout the search neighbourhood, will provide a more reliable estimate.

Multiple estimation runs were made using an octant search, with a minimum number of samples of 16 for Indicated and 8 for Inferred resource classification. A search range of 25m x25m x10m was used Indicated Resource categories and 37.5m x 37.5m x15m used for Inferred Resource category. Minimum octants filled were 4 for Indicated and 2 for Inferred.

Density values were assigned to the rocks masses depending on their oxidation state with transported material assigned 2.27, underlying oxidised material assigned 2.35, transitional material 2.50 and fresh material 2.65.

4.1.7 Cut-off Grade

The cut-off used for V2 gold is 0.35g/t gold and the cut-off used for A39 silver is 53g/t silver. A cut-off of 2.5g/t Au was used for the V2 underground resource.

4.1.8 Mining and Metallurgical Methods and parameters and other modifying factors considered to date

See sections 4.2.3 and 4.2.4 below.

4.2 Mt Carlton Ore Reserve

4.2.1 Material Assumptions for Ore Reserves

The Mt Carlton open pit Ore Reserve estimate is defined within a revised final pit design which is based on detailed geotechnical design parameters, practical mining considerations and mining depletion at 31 December 2013. The spatial constraint (final pit design) remains largely unchanged from that used last year. The updated Ore Reserve cost base assumptions are based on demonstrated performance with supported cost supported cost reduction initiatives and vary in line with changing activity levels at the site over the life of operation. The open pit Ore Reserves are defined using a block cut-off approach. Current operations at Mt Carlton involve open pit mining of the V2 & A39 orebodies by conventional excavator-truck operation.

4.2.2 Ore Reserve Classification

All of the in-situ Ore Reserves are currently derived from Indicated Resources. The only Probable Reserves derived from Measured Resources are those reported in known and quantified stockpiles.

4.2.3 Mining Method

Current open pit mining at Mt Carlton is a conventional truck and excavator operation, with standard waste rock dumps, ore stockpiling and stockpiling lower grade ore. This excavator fleet is utilised to selectively mine ore material and waste from a total 5m design bench height in two ‘flitches’ each of 2.5m height. Ore dilution and recovery loss is accounted for in this process and no additional mining dilution or recovery factors are applied to the Mt Carlton Open Pit Ore Reserve estimate. The current operations demonstrate the appropriateness of this mining method as the basis of the Ore Reserve estimate.

4.2.4 Processing method

The Ore Reserve estimate is predicated on the current 800Ktpa site based ore processing facilities currently exploiting the gold, silver and copper sulphide resources by flotation to produce a gold rich silver & copper concentrate from V2 and a silver rich & copper concentrate from A39. Concentrate is exported to smelter customers in China via Townsville port facilities. Crusher feed is a blend of several material types to ensure a steady feed state and minimise the effect of deleterious minerals such as arsenic and lead. Ore treatment and processing consist of crushing and grinding in a bulk sulphide flotation circuit to produce a polymetallic sulphide concentrate. Finer grind through ISA mills optimises gangue rejection to improve concentrate grade. Silver and gold concentrate production alternate in campaigns.

Mt Carlton open pit ore recoveries are dependent on ore type, material properties and grade. Metal recoveries for Mt Carlton open pit ore for the Ore Reserve estimate are based on historic production. The current and estimated future average recoveries at V2 are 89% for Au, 91% for Ag and 91% for Cu. The current and estimated future average recoveries at A39 for Ag is 88% and Cu is 92%.

Deleterious elements are not anticipated to impact on the value of concentrate produced due to blending of the product from the open pit.

4.2.5 Cut-off Grade

The marginal cut-off grade used to report the Ore Reserves is derived from the cost of processing ore (including site general and administration costs), additional incremental ore mining costs, metallurgical recoveries, royalties and gold price. A grade of 0.90g/t Au for the V2 pit and 53.0g/t Ag for A39 pit has been used for the Ore Reserve estimate.

4.2.6 Estimation Methodology

See section 4.1.5 above.

4.2.7 Material Modifying Factors

There are no concerning material modifying factors that need to be highlighted with the Ore Reserve. All regulatory leasing, approvals, licensing, agreements and current infrastructure are in place, which considers this estimation higher than that of a feasibility study.

5.0 MT RAWDON

The December 2013 Mt Rawdon Mineral Resource estimate of 51.1Mt at 0.8g/t gold for 1,234koz represents a decrease of 54koz net of mining depletion compared to the December 2012 estimate of 56.7Mt at 0.7g/t gold for 1,288koz. Changes are largely due to:

- Mining depletion during the period (-129koz)
- Increase due to decreased mining costs and subsequent adjustment of the optimisation shell (+104koz)
- Decrease due to geological re-interpretation and optimisation of estimation parameters (-18koz)

Mt Rawdon Mineral Resources - December 2013												
Mineral Resource	Measured			Indicated			Inferred			Total Resource		
	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)
Mt Rawdon	-	-	-	42.40	0.8	1,060	7.94	0.6	162	50.34	0.8	1,222
Stockpile	0.76	0.5	12	-	-	-	-	-	-	0.76	0.8	12
Total	**0.76**	**0.5**	**12**	**42.40**	**0.8**	**1,060**	**7.94**	**0.6**	**162**	**51.10**	**0.8**	**1,234**

Notes:
Data is reported to significant figures and differences may occur due to rounding
Mt Rawdon Mineral Resources have been reported above a cut-off grade of 0.23g/t gold and constrained to an A$1,800/oz pit optimisation shell

The December 2013 Mt Rawdon Ore Reserve estimate of 30.6Mt at 0.9g/t gold for 862koz represents a decrease of 164koz net of mining depletion compared to the December 2012 estimate of 39.8Mt at 0.8g/t gold for 1,026koz. Changes are largely due to mining depletion during the period (-158koz).

Mt Rawdon Ore Reserves - December 2013									
Ore Reserve	Proved			Probable			Total Reserve		
	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)
Mt Rawdon	-	-	-	29.80	0.9	850	29.80	0.9	850
Stockpile	0.76	0.5	12	-	-	-	0.76	0.5	12
Total	**0.76**	**0.5**	**12**	**29.80**	**0.9**	**850**	**30.56**	**0.9**	**862**

Notes:
Data is reported to significant figures and differences may occur due to rounding
Ore Reserves are reported above a 0.3g/t gold cut-off

5.1 Mt Rawdon Mineral Resources

5.1.1 Geology and Geological Interpretation

Alluvial gold was discovered at Mt Rawdon on Swindon Station in 1946. The current open pit has been producing continuously since 2001. Mt Rawdon is an intrusion related gold system (IRGS) hosted by Triassic rhyodacitic volcaniclastics and intrusive rocks of the Aranbanga Group. The large Eastern Dacite and offshoot Western Dacite intrude a thick water-deposited volcaniclastic pile, subsequently cut by a range of barren dykes and sills ranging from andesitic to rhyolitic composition. The original topographic high of Mt Rawdon was caused by an erosion-resistant suite of shallow north-dipping trachyte dykes.

Mineralisation is strongly associated with disseminated pyrite and small sulphide-rich veinlets controlled by structure and lithology. It forms a large steep-dipping massive low grade gold-silver orebody within an envelope of K-spar and strong sericite alteration. Internally the deposit contains numerous thin tabular high grade zones in multiple intersecting orientations. Another major mineralisation control is the sub-vertical western contact of the Eastern Dacite. The mineralisation is characterised by fresh sulphides, predominantly disseminated pyrite, and as a general rule the higher the sulphide content the higher the gold grade. Associated minor sulphides are chalcopyrite, sphalerite, galena and bismuth sulphosalts.

5.1.2 Sampling and Sub-sampling

All available drilling within the orebody was sampled. Earlier RC and DD holes were logged by a geologist and sampled at regular 2m intervals ignoring lithological contacts. In recent years however, all diamond holes are logged and sampled at intervals based on lithological contacts to a maximum length of 1.5m, minimum length 0.5m. Diamond drill core is routinely sampled as half core although some previous holes were whole cored. Core sample preparation process involved drying, crushing and pulverising to produce a pulped product with the minimum standard of >80% passing 75µm. RC drilling is used to obtain 1m samples from which a 3 – 5kg subsample is obtained using a cone or riffle splitter and pulverised to produce a charge for fire assaying for gold. Field duplicates were collected at a frequency of 1 in 30.

5.1.3 Drilling Techniques

There has been a long history of drilling at Mt Rawdon since 1979 with methods and processes evolving with industry practices. Current estimates are based on 529 drillholes which are mainly reverse circulation (RC) and/or NQ diameter diamond (DD) holes, with a small proportion of percussion holes in the upper mined portion of the orebody.

5.1.4 Sample Analysis Methods

All drill samples have historically been sample prepped and assayed at either the SGS Townsville or ALS Brisbane commercial laboratories. RC and diamond drilling samples were regularly analysed for gold (50g charge fire assay with AAS finish) and silver (3- acid digest followed by AAS finish), and some holes were also assayed for carbon and sulphur (LECO). Standard field QC procedures are routinely employed involving the submitting of certified reference materials, blanks and field duplicates. Since Evolution Mining ownership commenced in November 2011 the total QC sample insertion rate has been 11.6% (~1 in 9 samples).

5.1.5 Estimation Methodology

The gold grade estimate at Mt Rawdon for mineralised domains was performed using MIK with silver estimated using Ordinary Kriging interpolation methods. Gold and silver grades were estimated in parts per million (ppm) values. The deleterious elements carbon and sulphur were estimated as whole rock percentages to determine the potential acid forming potential (PAF). Carbon and sulphur were estimated using Inverse Distance cubed due to the limited data and precision requirements. The Mt Rawdon estimate was performed using Isatis™ software with post-model editing and validation performed using Micromine™ software.

The block model extents ranged between 374,610mE to 375,990mE, 7,203,410mN to 7,204,690mN and -445mRL to 285mRL with regular block size dimensions of 20m by 20m by 15m (X, Y and Z). The mine grid used to estimate the Mineral Resource is AGD Zone 55.

Waste domains were assigned the background grade value from the statistical evaluation of the composite dataset. The mineralised domain is separated as above and below the quartz-feldspar-biotite porphyry (QFBP) intrusive dyke restricted to an interpreted 0.1 g/t gold envelope.

The drill hole assay database was composited to 6m intervals using a 0.5m interval minimum length and assigned to a model domain matching domains in the Mineral Resource block model.

The gold MIK estimate used 14 indicator thresholds with no restriction imposed on the composited grades. The block model estimate parameters were reviewed and optimised using Kriging Neighbourhood Analysis (KNA):

- Block size
- Number of samples
- Search range
- Block discretisation

The silver OK estimate applied a cut-off threshold of 14g/t silver for ‘Above QFBP’ domain and 18g/t silver for ‘Below QFBP’ domain after 10m. Key gold and silver estimate parameters include:

- Quadrant search,
- Two passes with >80% block estimated in first pass for each domain,
- Pass 1 with minimum of 8 samples and maximum of 24 samples,
- Pass 2 with minimum of 6 samples and maximum of 12 samples,
- Search directions and ranges orientated to variography and mineralisation trend.
- Block discretisation of 3 by 3 by 2 (X, Y and Z).

A formal peer review and validation process was performed by Evolution personnel involving visual inspection of section views, statistical analysis and comparison of estimated grades to input composite grades, generation of Swathe plots, tonnes and grade charts which also compared the new estimate to the previous reported estimate.

Reconciliation performance and validation supports the accuracy of the global Mineral Resource estimate reported.

5.1.6 Resource Classification

The Mineral Resource classification is based on demonstrated geological and grade continuity and confidence in the grade estimation.

The classification approach incorporates a comprehensive and holistic approach of the following criteria with reference to data quality and continuity:

- Data type (i.e. hole type, drill hole spacing and orientation, sample type and assay method)
- Statistical performance of the estimate (i.e. Kriging Efficiency and slope of regression)
- Variography analysis
- Estimate parameters (i.e. number of samples, distance of samples, estimation technique)
- Visual inspection

Mineral Resources are assigned a numerical code in the Mineral Resource block model by the generation of interpreted 3 dimensional solids representing the Mineral Resource classification to be assigned.

5.1.7 Cut-off Grade

The Mineral Resource cut-off grade of 0.23g/t Au is based on the economic criteria reflecting Evolution’s mining (open-pit) and milling costs at Mt Rawdon. The cut-offs reflect the current and anticipated mining strategy and practices.

5.1.8 Mining and Metallurgical methods and parameters and other modifying factors considered to date

See sections 5.2.3 and 5.2.4 below.

5.2 Mt Rawdon Ore Reserve

5.2.1 Material Assumptions for Ore Reserves

The Mt Rawdon open pit Ore Reserve estimate is defined within a revised final pit design which is based on detailed geotechnical design parameters, practical mining considerations and mining depletion at 31 December 2013. Final pit designs have been developed from updated pit optimisation shells. The updated Ore Reserve cost base assumptions are based on demonstrated performance with supported cost supported cost reduction initiatives and vary in line with changing activity levels at the site over the life of operation. The open pit Ore Reserves are defined using a block cut-off approach. Current operations at Mt Rawdon involve open pit mining of the orebody by conventional excavator-truck operation.

5.2.2 Ore Reserve Classification

All of the in-situ Ore Reserves are currently derived from Indicated Resources. The only Probable Reserves derived from Measured Resources are those reported in known and quantified stockpiles.

5.2.3 Mining Method

Current open pit mining at Mt Rawdon is a conventional truck and excavator operation, with standard waste rock dumps, ore stockpiling and reclaim of lower grade ore. This excavator fleet is utilised to selectively mine ore material and waste from a total 15m design bench height in two 'flitches' each of 7.5m height. Ore dilution and recovery loss is accounted for in this process and no additional mining dilution or recovery factors are applied to the Mt Rawdon Open Pit Ore Reserve estimate. The current operations demonstrate the appropriateness of this mining method as the basis of the Ore Reserve estimate.

5.2.4 Processing Method

The Ore Reserve estimate is predicated on the current 3.5Mtpa site based ore processing facilities. An operating history of over 10 years supports the metallurgical parameters used in the Ore Reserve estimation.

5.2.5 Cut-off Grade

The marginal cut-off grade used to report the Ore Reserves is derived from the cost of processing ore (including site general and administration costs), additional incremental ore mining costs, metallurgical recoveries, royalties and gold price. A grade of 0.30g/t Au has been used for the Ore Reserve estimate.

5.2.6 Estimation Methodology

See section 5.1.5.

5.2.7 Material Modifying Factors

Mt Rawdon has operated continuously for over 13 years, since the mine and processing plant were opened in early 2001. Mt Rawdon is considered to be a mature operation with reliable historical data. Inputs for the Ore Reserve estimate are generally consistent with current and planned operating practices and experience. For this reason the analysis is considered to be at a higher level than a feasibility study.

Mining and ore processing operations at the Mt Rawdon open pit are conducted pursuant to a series of granted mining leases, exploration licences, general purpose leases and miscellaneous licences and associated environmental and other approvals. The granted tenements and permits cover all infrastructure in the immediate vicinity of the mine site, including the open pit, mill, waste rock dumps and tailings storage facilities.

6.0 TWIN HILLS

Twin Hills Mineral Resource was first disclosed under the JORC Code 2004 Edition and has not been updated. It is not related to a material mining project and has not materially changed since last reported. The December 2013 Mineral resource estimate of 4.62Mt at 2.7g/t gold for 399koz therefore remains unchanged from the December 2012 estimate.

However, a prudent reclassification of resources to the Inferred category was completed as the previous Mineral Resource estimate relies on historical data, and no work has been undertaken since 2009.

The Mineral Resource estimate for the Twin Hills open-pit deposit is reported within a A$1,500/oz optimisation pit. The Mineral resource estimates for the Twin Hills underground deposits are reported above a cut-off of 2.0g/t gold.

Twin Hills Mineral Resources - December 2013

Mineral Resource	Measured			Indicated			Inferred			Total Resource		
	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)	Tonnes (Mt)	Grade Au (g/t)	Cont. Metal Au (koz)
Open-Pit												
309 Deposit	-	-	-	-	-	-	3.06	2.1	204	3.06	2.1	204
Underground												
Lone Sister	-	-	-	-	-	-	1.02	3.7	120	1.02	3.7	120
309 Deposit	-	-	-	-	-	-	0.54	4.3	74	0.54	4.3	74
Total Hills	-	-	-	-	-	-	**4.62**	**2.7**	**399**	**4.62**	**2.7**	**399**

Notes:

Data is reported to significant figures and differences may occur due to rounding

Twin Hills Mineral Resources have been reported above a cut-off grade of 2.0g/t gold for underground, 0.5 g/t gold for open-pit and within a A$1,500/oz pit shell

Twin Hills Lone Sister was estimated using Ordinary Kriging and 309 using Multiple Indicator Kriging (E Type) into blocks with dimensions 5 metres east by 5 metres north by 5 metres elevation

Twin Hills is reported under the 2004 JORC Code as there has been no work undertaken on the resource since 2009 and the resource is based on historical data

APPENDIX 1: JORC CODE 2012 ASSESMENT AND REPORTING CRITERIA

The following information is provided in accordance with Table 1 of Appendix 5A of the JORC Code 2012 - Section 1 (Sampling Techniques and Data), Section 2 (Reporting of Exploration Results), Section 3 (Estimation and Reporting) and Section 4 (Estimation and Reporting of Ore Reserves).

CRACOW

JORC Code 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	Numerous sample types have been collected at Cracow and used in Mineral Resource estimates. Predominately these were Diamond Drill core, Rock Chip (hammer collection of development face samples) and Reverse Circulation (RC). A small number of samples from Trenches/Costeans were used in the Klondyke estimation and some of the earlier ore bodies. Sample intervals for drill core and face samples were determined by visual logging of lithology type, veining style/intensity and alteration style/intensity to ensure a representative sample was taken. In addition, sampling is completed across the full width of mineralisation. Minimum and maximum sample intervals were applied using this framework. RC samples were collected on 1m intervals. No instruments or tools requiring calibration were used as part of the sampling process. Industry standard procedures are followed and there is no significant coarse gold issue that affects the sampling protocols. Nominal 3kg samples from face sampling and drilling are subsampled to produce a 50g sample submitted for fire assay.
Drilling techniques	A combination of drilling techniques was used across the Cracow lodes. RC (face sampling bit), Diamond HQ/NQ (triple tube and standard) and LTK60 were the most commonly used. A small number of the HQ and NQ holes were orientated. Recording of the size of hole, or if the hole was drilled by diamond or RC techniques was sometimes missing in the older data. This uncertainty in the input data was considered when assigning resource categories to the blocks these particular holes informed. A summary of the different drill method techniques used across the Cracow ore bodies is summarised in the table below.

	Drill Method							
Ore Body	**Surface DD Unknown Size**	**Surface DD HQ or NQ**	**Surface RC**	**Surface Unknown Hole Type**	**Underground DD NQ or LTK60**	**Underground Face**	**Total Holes**	**Total Holes & Faces**
Kilkenny & Tipperary	2	93	0	0	466	1095	561	1656
Roses Pride	19	27	10	78	100	695	234	929
Phoenix	10	0	0	0	44	292	54	346
Empire	20	10	0	0	120	179	150	329
Griffin	9	0	0	0	32	2	41	43
Klondyke	45	0	43	40	119	133	247	380
Coronation	3	23	0	0	15	0	41	41

Criteria	Commentary
Drill sample recovery	Drill core – the measurement of length drilled compared to length of core recovered was completed for each drilled run by the drill crew. This was recorded on a core loss block placed in the core tray for any loss identified. Marking up of the core by the geological team then checked and confirmed these core blocks, and any additional core loss was recorded and blocks inserted to ensure this data was captured. Any areas containing core loss were logged using the lithology code "Core Loss" in the lithology field of the database. RC Chip Samples – RC samples weren't weighed at Cracow, so a determination on sample recovery wasn't completed. The drill crew recorded any underground voids they encountered to ensure lack of sample return wasn't confused with sample loss. These areas were coded "Void" in

Criteria	Commentary
	the lithology field of the database. Due to the small amount of samples that the RC samples contributed to the resource estimations at Cracow, this approach to sample recovery assessment is considered sufficient. Sample recovery loss at Cracow was calculated at less than 1% and wasn't considered an issue. Washing away of sample by the drilling fluid in clay or fault gouge material is the main cause of sample loss. In areas identified as having lithologies susceptible to sample loss, drilling practices and down-hole fluids were modified to reduce or eliminate sample loss. The drilling contract used at Cracow states for any given run, a level of recovery is required otherwise financial penalties are applied to the drill contractor. This ensures sample recovery is prioritised along with production performance. Mineralisation at Cracow was within quartz-carbonate fissure veins, and therefore sample loss rarely occurs in lode material. No relationship between sample recovery and grade was observed.
Logging	Geological logging was undertaken onsite by Evolution employees and less frequently by external contractors. Logging was completed using *LogChief* Software and uploaded directly to the database. A standard for logging at Cracow was set by the Core Logging Procedure *Cracow Procedures Manual 3rd Edition*. Drill Core is logged recording lithology, alteration, veining, mineral sulphides and geotechnical data. RC chip logging captured the same data with the exclusion of geotechnical information. Some historical data used at Cracow did not include lithological or geotechnical data. These holes are from Klondyke (35% of data) and Roses Pride (17% of data) lodes. Resource categorisation takes into account the quality and quantity of the data logged. Logging was qualitative. All drill core, RC chips and underground faces that were sampled during 2013 were photographed. Core and RC chips were photography photographed wet using a camera stand and an information board to ensure a consistent standard of photography and relevant information was captured. All core and RC chip samples collected were fully logged, except those previously noted at Klondyke and Roses Pride.
Sub-sampling techniques and sample preparation	Surface and underground drill core, was halved using an automatic core saw, with one half dispatched for analysis and the other half retained. All underground LTK60 was whole core sampled, with a small number of underground NQ holes whole core sampled during 2013. During 2013 the practice on site for RC samples was for a 7-1 split to be taken at the drill rig using a riffle splitter. The moisture condition of the sample was not captured. Given the small proportion of RC samples used in the Mineral Resource (1% of the Roses Pride data and 11% of the Klondyke data) this was considered acceptable. Whole/half core samples were crushed in a jaw crusher to > 70% passing 2mm; half of this material was split with a riffle splitter for pulverising. No RC samples required crushing in the jaw crusher. Core and RC samples were pulverised for 10-14 minutes in a LM5 bowl with a target of 85% passing 75µm. Grind checks were undertaken nominally every 20 samples. From this material approximately 120g was scooped for further analysis and the remaining material re-bagged. Duplicates were performed on batches processed by ALS every 20 samples at both the crushing and pulverising stages. This sample preparation for drill samples is considered appropriate for the style of mineralisation at Cracow. Sample prep for rock chip face samples was conducted at the Cracow onsite laboratory. Samples were crushed in a Jaw Crusher to 100% passing 5mm, this material was then split with a riffle splitter and pulverized for 4 minutes in a LM2 bowl with a target of 85% passing 75 µm. From this 400g was collected with a 150 (scoop) and packaged for transport to ALS Townsville. Duplicates were performed on batches processed by ALS Brisbane every 20 samples at both the crushing and pulverising stages. Grind checks were undertaken nominally every 20 samples, to ensure sample grind target of 85% passing 75µm was met. Duplicates were completed every 20 samples at both the crushing and pulverising stages, and no bias was found at any sub-sampling stage. Drill core was not orientated prior to cutting, as sample bias from non-orientation of core is considered minimal in respect to mineralisation at Cracow. Drill Core – for half core samples, on occasion the remaining core was quarter core sampled for confirmation of assay results. This was either sent to the same laboratory that assayed the original half core sample or to a different umpire laboratory. The majority of samples were whole core

Criteria	Commentary
	sampled, to ensure the entire sample stream was cut to give the most representative drill sample possible. Traditionally this practice of quarter coring decreases as the individual ore bodies mature and results indicated that the sub-sampling of the whole core is appropriate for the Cracow lodes. RC - Field duplicates were collected directly from the splitter every 20 samples The sample size collected is considered to be appropriate for the size and deportment of the gold mineralisation being sampled.
Quality of assay data and laboratory tests	Sample Analyses - The samples were analysed by 50g Fire Assay for Au with Atomic Absorption (AAS) finish and was performed at ALS Townsville. For Ag an Aqua Regia digest with AAS finish was completed, also at ALS Townsville. An analytical duplicate was also performed every 20 samples, aligned in sequence with the crushing and pulverising duplicates. The Fire Assay Method is a total technique. No other instruments that required calibration were used for analysis to compliment the assaying at Cracow. Fifteen externally Certified standards at a suitable range of gold grades (including blanks) were inserted at a minimum rate of 1:20 with each sample submission. All non-conforming results were investigated and verified prior to acceptance of the assay data. Results that did not conform to the QAQC protocols were not used in resource estimations. Monthly QAQC reports were produced to watch for any trends or issues with bias, precision and accuracy.
Verification of sampling and assaying	The pulverised material from 13 drill holes from the Griffin, Empire and Kilkenny ore bodies was sent to SGS Townsville during 2012 to test the results produced by ALS Townsville. Only a few samples fell outside the 10% error margin The umpire sampling confirmed the accuracy of the ALS Townsville assaying within acceptable error limits. The drilling of twin holes has not been common practice at Cracow. Twin holes that have been drilled show the tenor of mineralisation within the reportable domains were consistent between twin holes. All sample information was stored using Datashed, an SQL database. The software contains a number of features to ensure data integrity. These include (but not limited to) not allowing overlapping sample intervals, restrictions on entered into certain fields and restrictions on what actions can be performed in the database based on the individual user. Data entry to Datashed was undertaken through a combination of site specific electronic data-entry sheets, synchronisation from Logchief and upload of .csv files No adjustments are made to the finalised assay data received from the laboratory.
Location of data points	The position of surface holes was determined by differential GPS or handheld GPS. Underground drill-hole positions were determined by traversing using Leica TS15 Viva survey instrument (theodolite) in the local Klondyke Mine Grid. Down-hole surveys were captured by an Eastman camera for older holes and a Reflex camera on recent holes. The underground development face sample positions were determined by the distance (measured from a laser-distometer) to the face from a surveyed point in the drive. Mine workings (drives and stopes) used for resource depletions were surveyed using either the Leica TS15 Viva or an Optek Cavity Monitoring System (CMS) for stopes. The mine co-ordinate system at Cracow is named the Klondyke Mine Grid, which transforms to MGA94 Grid and was created and maintained by onsite registered surveyors. The Roses Pride mineralisation is located in close proximity to the surface, requiring a Topography wireframe/dtm. The topography wireframe was generated by the Survey Department from Airborne Laser Scan and ground surveying methods.
Data spacing and distribution	Exploration results are not being reported Sample spacing and distribution was deemed sufficient for resource estimation. Spacing and distribution varied from closely spaced 4m x 16m face samples in ore drives, through to a range of drill patterns: 20m x 20m, 40m x 40m and 80m x 80m.

Criteria	Commentary
	The sample spacing required for the resource category of each ore body is unique and may not fit the idealised spacing indicated above. All datasets were composited prior to estimation. The most frequent interval length was 1 metre, particularly inside and around mineralised zones. Sample intervals for most domains were composited to 1m, with a maximum sample length of no greater than 1.5m and a minimum sample interval of 0.2m. In some cases due to the narrow width of the domain, a compositing interval was selected to produce a single composite across the domain width.
Orientation of data in relation to geological structure	Sample bias from non-orientation of core is considered minimal in respect to mineralisation at Cracow. Not all core was orientated prior to cutting; however, core that was orientated was cut vertically along the bottom of the hole as indicated by the orientation line. Drill holes were designed to ensure angles of sample intersection with the mineralisation were as perpendicular as possible. Where a poor intersection angle of individual holes locally distorted the interpreted mineralisation, these holes may not have been used to generate the wireframe. The grade from these holes was on most occasions still used in the estimation, by "hardcoding" the domain code to the drill-hole file. Any bias that was introduced by these holes were contained by the estimation and search ellipse parameters; however, in extreme cases holes were removed from the estimation completely.
Sample security	Police Clearances are obtained for all staff and they are instructed on relevant JORC 2012 requirements and assaying is completed by registered laboratories. The core was transported by a private contractor by truck to the assay laboratories.
Audits or reviews	A site lab inspection was conducted by Evolution Geology Staff on the 28th May 2013. ALS Virginia Prep Lab and ALS Townsville Fire Assay lab inspection carried out by Evolution Staff (Chris Wilson) on 25th September 2013. The Cracow Datashed database was reviewed by Evolution database specialists during 2013.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	ML3219, ML3221, ML3223, ML3224, ML3227, ML3228, ML3229, ML3230, ML3231, ML3232, ML3243, ML80024, ML80088, ML80089, ML80114, ML80120, ML80144 and EPM15981 are all wholly owned by Newcrest Operations Limited. With the formation of Evolution Mining, the tenement titles are in the process of being transferred to Evolution Mining's wholly owned subsidiary, Lion Mining Pty Ltd. All tenure is current and in good standing.
Exploration done by other parties	The Cracow Goldfields were discovered in 1932, with the identification of mineralisation at Dawn then Golden Plateau in the eastern portion of the field. From 1932 to 1992 mining of Golden Plateau and associated trends produced 850koz. Exploration across the fields and nearby regions was completed by several identities including BP Minerals Australia, Australian Gold Resources Ltd, ACM Operations Pty Ltd, Sedimentary Holdings NL and Zapopan NL In 1995, Newcrest Mining Ltd (NML) entered into a 70% share of the Cracow Joint Venture. Initially exploration was targeting porphyry type mineralisation, focusing on the large areas of alteration at Fernyside and Myles Corridor. This focus shifted to epithermal exploration of the western portion of the field, after the discovery of the Vera Mineralisation at Pajingo, which shared similarities with Cracow. The Royal epithermal mineralisation was discovered in 1998, with further discoveries of Crown, Sovereign, Empire, Phoenix, Kilkenny and Tipperary made from 1998 up to 2008. Evolution was formed from the divestment of Newcrest assets (including Cracow) and the merging of Conquest and Catalpa in 2012. Evolution continued exploration at Cracow from 2012.
Geology	The Cracow project area gold deposits are in the Lower Permian Camboon Andesite on the south-eastern flank of the Bowen Basin. The regional strike is north-northwest and the dip 20° west-southwest. The Camboon Andesite consists of andesitic and basaltic lava, with agglomerate, tuff and some inter-bedded trachytic volcanics. The andesitic lavas are typically porphyritic, with

Criteria	Commentary
	phenocrysts of plagioclase feldspar (oligoclase or andesine) and less commonly augite. To the west, the Camboon Andesite is overlain with an interpreted disconformity by fossiliferous limestone of the Buffel Formation. It is unconformably underlain to the east by the Torsdale Beds, which consist of rhyolitic and dacitic lavas and pyroclastics with inter-bedded trachytic and andesitic volcanics, sandstone, siltstone, and conglomerate. Mineralisation is hosted in steeply dipping low sulphidation epithermal veins. These veins found as discrete and as stockwork and are composed of quartz, carbonate and adularia, with varying percentages of each mineral. Vein textures include banding (colloform, crustiform, cockade, moss), breccia channels and massive quartz, and indicate depth within the epithermal system. Sulphide percentage in the veins are generally low (<3%) primarily composed of pyrite, with minor occurrences of hessite, sphalerite and galena. Rare chalcopyrite, arsenopyrite and bornite can also be found. Alteration of the country rock can be extensive and zone from the central veined structure. This alteration consists of silicification, phyllic alteration (silica, sericite and other clay minerals) and argillic alteration in the inner zone, grading outwards to potassic (adularia) then an outer propylitic zone. Gold is very fined grained and found predominantly as electrum but less common within clots of pyrite.
Drill hole Information	No exploration results has been reported in this release, therefore no drill hole information to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to drill hole information relevant to the Mineral Resource estimate can be found in Section 1 – "Sampling techniques" and "Drill sample recovery."
Data aggregation methods	No exploration results has been reported in this release, therefore there are no drill hole intercepts to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to data aggregation methods relevant to the Mineral Resource estimate can be found in Section 1 – "Sampling techniques" and "Drill sample recovery."
Relationship between mineralisation widths and intercept lengths	No exploration results has been reported in this release, therefore there are no relationships between mineralisation widths and intercept lengths to report. This is not relevant to this report on Mineral Resources and Ore Reserves.
Diagrams	No exploration results has been reported in this release, therefore no exploration diagrams have been produced. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Balanced reporting	No exploration results has been reported in this release, therefore there are no results to report. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Other substantive exploration data	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Further work	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	All sample data used in the estimation was stored in the site Datashed database. User groups were assigned for various staff, dictating what changes to the database can be made. Restricted access was in place for most of these users to ensure that any changes were controlled. The site Datashed database has several validation checks. For example no overlapping data intervals, no duplicate records, collar surveys required, data lengths cannot exceed maximum hole depth and sample numbers from an assay file must match entirely sample numbers of a drill hole. All holes and face samples are checked for correct collar coordinates, down hole surveys and excessive down-hole deviations. During resource wireframe interpretation, holes were checked against surrounding holes to confirm

Criteria	Commentary
	geology logging and assay values. All holes and faces are photographed, to confirm correct geology logging and sample assays.
Site visits	The Competent Person is an employee of Evolution Mining Limited and has been a rostered staff member on-site at Cracow.
Geological interpretation	The low sulphidation epithermal veins of the western portion of the Cracow Field have been mined since 2005. Extensive mapping and modelling of development was undertaken from the commencement of mining and was incorporated into current geological interpretation. Controls and orientation of most of the different lodes mineralisation are well understood; however, in cases of geological uncertainty, this was reflected in the resource classification assigned to the area of the resource model. Geological surfaces were interpreted using a combination of drill-hole and face sampling data and underground mapping lines. Three dimensional surfaces were created using Datamine v3 software. As the Cracow mineralisation occurs in discrete structures, any change in either the interpreted orientation or grade continuity would have impact on the estimation methodology and the resulting estimate. No alternative interpretation of the mineralisation style or geometry was considered for Cracow. As the mineralisation at Cracow is hosted by discrete structures, geology (lithology & vein percent) along with Au Grade was the principle controls for domaining, and strongly influenced the estimation. Ore bodies were domained, and in some cases sub-domained, into various lithology-grade domains. During estimation, blocks within a certain domain were only able to estimate using the appropriately flagged sample points. Gold mineralisation at Cracow is located in shear hosted quartz/carbonate veining, with low grade mineralisation in the wall rock. At Cracow veins are found only in andesite lava due to its brittle fracture qualities. Small scale lateral and vertical offsetting by faults has been observed at various locations. Rhyolite (rarely mineralised) and barren mafic dykes were recorded intruding and offsetting the veins.
Dimensions	The extents and variability of the mineral resource veins is given in the table below:

	Ore Body Extents				Resource Extents (Inferred)			
Ore Body	**Length (m)**	**Height (m)**	**Thickness (m)**	**Depth Below Surface (m)**	**Length (m)**	**Height (m)**	**Mean Thickness (m)**	**Thickness Variability (m)**
Kilkenny /Tipperary	900	600	1-10	455	850	425	2.9	1.0-6
Roses Pride	900	250	1-6	10	600	230	1.3	0.5-3
Phoenix	300	300	1-6	420	601	140	1.8	0.6-3
Empire	600	300	1-5	350	450	210	1.4	0.5-3
Griffin	350	300	1-4	360	220	240	1.1	0.6-2
Klondyke	450	350	1-5	0	450	280	1.7	1.0-3
Coronation	350	500	2-10	480	160	170	3.9	2.0-6
Sovereign	500	350	1-8	140	350	270	4	1.5-6

Estimation and modelling techniques	The estimations were performed using Datamine V3 software. Ordinary Kriging was the preferred method of estimation used for Cracow resources. In some cases other estimation techniques such as Inverse Distance were used. Variograms were generated using the composited drill-hole file in Snowdens Supervisor V7. Search ellipses were orientated with the grade continuity as identified by the variography. The treatment of extreme grade values is explained below (top cuts). Domaining criteria are discussed in the Geological Interpretation section above. Previous estimations of Cracow resources were compared against new models to measure the effect of additional data and changes in estimation parameters

Criteria	Commentary
	Comparisons between reconciled mine production and previous models were completed on a monthly basis. Any issues identified with this comparison were taken into account during subsequent resource updates. Ag is estimated with Au as a by-product in the sale of gold doré, and is calculate estimated from its own data. No deleterious elements were estimated or assumed. Generally, block size is half of the drill spacing. The average drill spacing and block size used for each ore body is summarised below.

Ore Body	Average Drill Spacing (m)	Block Size X (m)	Block Size Y (m)	Block Size Z (m)
Kilkenny/ Tipperary	12.5 x 16	5.0	5.0	10.0
Roses Pride	20 x 20	5.0	10.0	10.0
Phoenix	30 x 30	20.0	10.0	10.0
Empire	20 x 20	5.0	10.0	10.0
Griffin	30 x 30	5.0	10.0	10.0
Klondyke	20 x 20	2.0	12.5	10.0
Coronation	40 x 40	20.0	20.0	20.0

Criteria	Commentary
	No selective mining units were assumed in this estimate. A correlation was noted between Au and Ag grades; however it's not used in the resource estimate. Blocks were generated in between the hanging-wall and footwall wireframe surfaces that defined each domain. Blocks within these domains were estimated using sample points located within the same domain. On occasion a block was allowed to estimate using samples for a limited distance across a domain boundary. This was most common when sub-domaining within a particular structure. Top cuts were applied to the data to control the influence of high grade Au and Ag values, interpreted to be not representative of the mineralisation. A combination of Log probability plots, % step change in grades and the Co-efficient of Variation (CV) was used to determine top-cut values. The effect of the applied top-cuts was reviewed in respect of the mean and CV of the data The model was validated by comparing statistics of the estimated block grade against the composite sample data, visual inspection in Datamine of block grades to drill-hole grades in plan/sectional views, and using Swath Plots. The model was also reconciled against production data.
Moisture	All estimations were performed on a "Dry" basis.
Cut-off parameters	Based on mining assumptions and life of mine (LOM) the indicative cut-off grade for reporting purposes is 2.8g/t Au.
Mining factors or assumptions	Mining of the Cracow ore bodies commenced in 2004. The mining method adopted is widely termed Modified Avoca. This is where stopes are extracted between levels based on geotechnical recommended parameters for stope lengths and heights, then backfilled with loose or consolidated fill before the next retreating stope is extracted. The method has been used extensively at Cracow throughout its ten year mine life. All deposits estimated in this report are amenable to this mining method.
Metallurgical factors or assumptions	The ore is to be processed through a traditional CIP/ CIL process plant at a current rate of approximately 550ktpa. The current and estimated future recoveries for gold are 94.0% and 80.0% for silver. An operating history of around ten years supports the metallurgical parameters used.
Environmental factors or assumptions	The majority of waste rock is consumed underground as loose rock backfill of mined stopes. Waste rock from development for use in building and extensions of tailings dams was sampled once brought to surface, and the acid potential of the material determined. Due to the low sulphide content and carbonate alteration of the barren andesite used for construction, the potential for acid

Criteria	Commentary
	mine drainage is minimal.
Bulk density	A combination of assumed and determined bulk density was used across the various resource models at Cracow. Collection of density data from drill core was completed from 2012. Most lodes had an adequate number of density samples, but some require a level of assumption. Given the lithological similarities between the discrete mineralised lodes at Cracow and reconciliation with mine production this is deemed acceptable. Bulk density measurements taken from 2012 were calculated using a non-wax coated water immersion method. Testing to determine the suitability of density method comparing wax coated, non-wax coated and picnometer was completed, with non-wax coated deemed appropriate. All deposits except Golden Plateau occur in “fresh” rock, and a single bulk density is applied within each domain based on the samples collected. Differences in density between lode, halo and country rock were noted and designated as appropriate. An oxidised and non-oxidised density was applied at Golden Plateau above or below a wireframe surface. Little variation in density values within a domained lode was noted and a single density value was calculated and assumed for each domain.
Classification	Various drill space patterns were used for the same resource classifications across separate lodes, due to comparative differences of the resource models. Resource categorisation was based on the confidence of the model, dependent but not limited to complexities relating to vein geometry, assay variability and faulting. The assigning of resource classification was based primarily on a combination of drilling density and which search volume used to estimate the block. Numerous factors related to the reliability in the quality of the sample data or the confidence of the geological interpretation, are also considered when assigning the resource classification. The Competent Person considers the applied resource classifications to be appropriate.
Audits or reviews	A review of the Mineral Resource estimates and processes was undertaken by an independent consultant in February 2014.
Discussion of relative accuracy/ confidence	The relative accuracy of the Mineral Resource estimate reflects the classification applied to the mineral resource. Reconciliation of the mineral resource estimate against production supports the classification. The relative accuracy relates to the global mineral resource estimate. Over the life of the Cracow Project estimated grades have performed well against mill recovery, with a slight positive reconciliation. The last 12 months of data comparing reconciled production to the mined December 2012 resource model is shown in the table below.

Reconciled Production				
Ore Body	**Tonnes (t)**	**Grade (g/t)**	**Gold (oz)**	
Crown	41,744	9.22	12,378	
Empire	47,638	4.93	7,554	
Kilkenny/ Tipperary	246,500	5.50	43,613	
Phoenix	39,952	6.23	8,000	
Roses Pride	130,969	6.06	25,511	
Total	**506,803**	**5.96**	**97,056**	
2012 Resource Model	**Tonnes (t)**	**Grade (g/t)**	**Gold (oz)**	
Crown	35,967	12.01	13,885	
Empire	15,303	14.82	7,293	
Kilkenny /Tipperary	158,555	8.13	41,457	
Phoenix	26,550	9.31	7,945	
Roses Pride	65,895	10.04	21,264	
Total	**302,270**	**9.45**	**91,844**	
Difference	**Tonnes (t)**	**Ounces (oz)**	**Tonnes (%)**	**Grade (%)**

Criteria	Commentary

Crown	-5,777	1,507	-16%	11%
Empire	-32,335	-261	-211%	-4%
Kilkenny /Tipperary	-87,945	-2,156	-55%	-5%
Phoenix	-13,401	-55	-50%	-1%
Roses Pride	-65,075	-4,247	-99%	-20%
Total	**-204,533**	**-5,211**	**-68%**	**-6%**

The 200Kt additional tonnes mined (shown in the table above) were from dilution. Open stope mining of Kilkenny/Tipperary, Crown and Phoenix resulted in over break of the low grade halo of mineralisation, diluting the ore material. Dilution issues for the Empire and Roses Pride Lodes were from the 5m wide development of the veins, which during this period was as narrow as 1m wide. In addition, stoping of these narrow structures at Roses Pride produced similar diluted results.

The mining techniques at Cracow are to be reviewed during the next reporting periods to determine ways to reduce dilution.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	The estimation of Mineral Resources is outlined in Section 3. All Measured and Indicated resource classifications where considered for conversion to Ore Reserves. Mineral Resources are reported inclusive of Ore Reserves.
Site Visits	Regular site visits have been completed throughout the year to gain an understanding of the geological setting, mining methods and mining factors applied to the Ore Reserve.
Study Status	The study to convert Mineral Resources to Ore Reserve is an operational design study. Pajingo is considered to be a mature operation with around 10 years of historical data.
Cut-off parameters	Cut-off grades are not used to estimate Ore Reserve, they are more a generalisation of economic areas. There are numerous cut off values dependent on cost structures applied. A fully costed indicative stoping cut-off grade of 3.5g/t is representative of a mine cut-off grade. All reserves are fully costed within an economic model and based on the proportion of operation and/or capital development required for ore extraction. Thus the cut-off grade varies dependent on these factors, and no one cut-off grade has been used for the Ore Reserve estimation.
Mining factors or assumptions	Mineable panels have been created based on typical level intervals for the Modified Avoca mining method currently used at the operation. The Mineral Resource models have been divided into stope panels 20m long at level interval height (15-20m). No geotechnical evaluation of the Ore Reserve has been undertaken, however geotechnical parameters are based on current stoping practices at Cracow. All stopes panels have a minimum stoping width of 1.8m. Dilution skins of 0.5m to the hanging wall and footwall are applied to the Kilkenny and Tipperary lodes, all other lodes have had 0.4m added. grade of 0.0g/t has been applied to all stopes. An additional 5% dilution is applied for operational recoveries of the ore. The current underground infrastructure is suitable to support the mining method with extensions to existing capital development.
Metallurgical factors or assumptions	The ore is to be processed through a traditional CIP/CIL process plant. The current and estimated future recoveries for gold are 94% and 80% for silver. An operating history of around 10 years supports the metallurgical parameters used in the Ore Reserve estimation.
Environmental factors or assumptions	Cracow is current with all environmental approvals and compliant to those conditions set out in such approvals.

Criteria	Commentary
Infrastructure	The mine is currently in operation and therefore has adequate infrastructure to support current and future operation.
Costs	Unit costs were derived from FY2014 (July 2013 to February 2014) actual costs. Throughout this period there was some one-off costs due to site restructure that were excluded from the cost estimates. Unit costs were calculated and applied to the economic assessment of the potential Ore Reserves these were; The unit cost estimates included leasing on underground equipment, no other site sustaining costs were used. State royalties are 5%.
Revenue factors	Revenue is calculated using a gold price A$1,350/oz. No revenue has been created from silver by-products due to the uncertain modelling of these grades in the Ore Reserve process.
Market assessment	Gold and silver at spot price.
Economic	The Ore Reserves have been economically evaluated through a standard financial model. All operating and capital costs as well as revenue factors were included in the financial model. This process has demonstrated that the Ore Reserves for the underground operations having a positive NPV.
Social	Currently Evolution Mining has agreements with Traditional Owners and on good terms with neighbouring pastoralists.
Classification	Classification of each stope panel is assessed on the proportion of tonnage in each resource classification. If the tonnage in the stope panel is greater than 50% Measured then it is classified as a Proved Ore Reserve. For stope panels greater than 50% Measured and Indicated it is classified as a Probable Ore Reserve. Stope panels that have greater than 50% of Inferred material are excluded from the Ore Reserve estimate. This process leads to some Inferred Resources being included in the Ore Reserve estimate as they will be mined as part of the stope panels. This amount is less than 5% of the Ore Reserves estimate.
Audits or reviews	Internal peer review by Evolution personnel has been conducted in accordance with Evolution's standards which confirms the stated Ore Reserve and supports the estimation parameters applied. This Ore Reserve has not been audited externally.
Discussion of relative accuracy/ confidence	The accuracy of the estimates within this Ore Reserve are mostly determined by the order of accuracy associated with the Mineral Resource model, the metallurgical input and the long term cost adjustment factors used. In the opinion of the Competent Person, the modifying factors and long term cost assumptions used in the Ore Reserve estimate are reasonable.

PAJINGO

JORC Code 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	The mineralised lodes of the Pajingo deposit have been defined through a combination of surface diamond drilling and RC drilling followed by underground diamond drilling and face sampling. Surface drilling has typically been from south to north with an average dip of 60°. Reverse circulation drilling was generally used to obtain 1m samples, each interval was logged by the geologist before determining intervals for analysis. A 2kg – 5kg sub-sample of the selected individual composited sample intervals were obtained using a spear, and more recently a rig mounted cone splitter or riffle were used. The sub-samples were pulverised by the assaying laboratory to produce a 30g charge for fire assaying for gold. Surface diamond drill core was logged by the geologist who subsequently determines the required sample intervals. Most surface diamond drill core was sampled as half-core with a minimum sample interval of 0.2m and maximum sample interval of 1.5m. Core samples were submitted to the laboratory for preparation and fire assay reporting gold and silver values by fire assay and up to 50 additional elements by Inductively Coupled Plasma Optical Emission Spectrometry. Sampling of underground diamond drillholes followed the same protocol as surface drilling up to October 2013 after which whole core samples of nominal 1m length were submitted. Underground drillholes were assayed for gold and silver by fire assay. Face sampling of underground development drives was routinely carried out as development advanced at 4m intervals, wall samples have also been taken where development has intersected mineralisation. Face and wall sampling involves a map being drawn and sample intervals determined bounded by lithology and alteration contacts (0.2m – 2m intervals). The geologist marks the contacts and/or sample intervals with paint and collects chips from within the interval directly into the sample bag. Prior to October 2013, face and wall samples were submitted for sample preparation and gold and silver analysis by fire assay. Samples were subsequently assayed by aqua regia for gold only. The location of drillhole collars were picked up and down hole surveyed by surveyors on surface using RTK (Real Time Kinetic) GPS and underground using TST (Total Station Tools).
Drilling techniques	Drilling at Pajingo is recorded dating to 1984. Third party specialised drilling contractors have been engaged to complete drill programs, the work methods, protocols and standards were consistent industry practice. Reverse circulation and diamond drilling methods have been employed at Pajingo. Surface holes were typically a RC collar to a depth of up to 400m often with a diamond drillhole tail to a maximum depth of 1500m. Reverse circulation holes were typically drilled with a 140mm/5.5 inch diameter bit. HQ/96mm diameter holes were drilled from surface and commonly reduced to NQ/60mm diameter holes at depth. Underground diamond drillholes were typically 60mm in diameter; employing both wireline (NQ) and conventional drilling (LTK60) methods. 95% of underground drillholes were less than 300m in length. A small number of longer diamond holes have been drilled underground with a maximum length of 850m. Underground face samples were taken as mining progressed in ore development drives, typically at 4m intervals. The drillhole represents a horizontal line of sampling (nominally 1.5m above the floor) across the exposed ore body and adjacent material.
Drill sample recovery	Recovery of surface and underground diamond core was recorded with the collection of geotechnical data, recovery has been determined based on core length compared to run length which is consistent with industry practice. Recovery has also been indirectly recorded with the qualitative geological data as “core loss”. Overall, diamond core recovery exceeds 95%. Recovery of RC drillholes has not been recorded consistently. A recovery and grade correlation study has not been completed with regard to recovery of RC drillholes.
Logging	Diamond and RC drill holes were qualitatively geologically logged for lithology, alteration, structure and veining. The level of detail recorded in the geological logging adequately supports the Mineral

Criteria	Commentary
	Resource estimation and related studies. The recording and storing of geological logs has evolved over time reflecting technology improvements & industry norms. The individual logs were stored electronically then uploaded to a central geological database. Geological logging information was available in the acQuire database for 97% of drillholes & 98% of face samples. Drill core and chip trays were routinely photographed and printed to 2005 then digitally photographed and stored to present. Remaining core is stored on-site and available for review.
Sub-sampling techniques and sample preparation	RC was generally used to obtain 1m samples; each interval was logged by the geologist before determining intervals for analysis. The samples selected for assaying were dried before a 2kg – 5kg subsample was taken at the drill site using a spear. Rig mounted cone splitter or riffle splitters producing a one eighth split were used for RC holes drilled since 2012. Preliminary composite samples were collect using the spear method. The subsample was sent to the assaying laboratory where it was dried, split using a riffle splitter and pulverised to a grind size of 85% passing 75µm. Field duplicates for RC samples were taken at a ratio of 1:20 and showed a good correlation to primary assays. Diamond drill core was logged by the geologist who subsequently determines the required sample intervals. Most surface diamond drill core was sampled as half-core with a minimum sample interval of 0.2m and maximum sample interval of 1.5m. Core samples were submitted to the assaying laboratory where they were dried, coarse crushed to around 10mm and then pulverised to 85% passing 75µm. Subsamples were typically less than 3kg which allowed the total subsample to be prepared and pulverised. Quarter core field duplicates for surface diamond holes were based on at a ratio of 1:20 and showed a good correlation to primary assays. Quarter core and half core field duplicates were inserted with underground diamond holes at a ratio of 1:20 and show a good correlation to primary assays. A small number of diamond drillholes have been sampled as whole core samples (after October 2013). The assaying laboratory dried, coarse crushed to ~10mm, split if >3kg and pulverised to 85% passing 75µm. Field duplicates were not submitted with whole core samples. Underground face samples were taken as mining progressed in ore development drives, typically at 4m intervals. The face sample category also includes wall samples that were taken in the same way in in areas where the development drive intersect the ore body. The drillhole represents a horizontal line of sampling (nominally 1.5m above the floor) across the exposed ore body and adjacent material. The sampling protocol is consistent with industry practice whereby the face is mapped, sample intervals are determined and marked from which the samples were collected. The sample interval is bounded by lithology and alteration contacts (0.2m – 2m intervals), where no boundaries were present a 1m nominal sample width is assigned. The geologist marks the contacts and/or sample intervals with paint and assigns a unique sample identifier to each interval corresponding to the sample bag. A geological hammer is used to loosen material from the face at multiple points within the interval which is then collected directly into the sample bag. Underground face samples were submitted to the assaying laboratory to be dried, coarse crushed to ~10mm, split if >3kg and pulverised to 85% passing 75µm. Field duplicates have been submitted at a ratio of 1 in 25 faces. A check on the minimum standard of 85% passing 75µm occurred in most sample batches, large batches of samples contained more than one check typically at a 1:50 ratio.
Quality of assay data and laboratory tests	Assay quality controls were consistent with industry practice. The assaying laboratories had internal systems and checks in place including the routine analysis of reference materials and lab duplicates. Additional certified reference materials (standards) and coarse blanks were submitted at a ratio of 1:20 with diamond core, RC chips and face chip samples and more frequently in ore zones. The performance of standards and blanks were reviewed for each batch, unexpected results were investigated and typically resolved with re-assays. All assays were reviewed by batch and flagged in the geological database as accepted, pending or rejected. Only accepted assays were used in the Mineral Resource estimation. The performance of standards over time was reviewed and no significant bias was observed. The supervising geologist inspected the laboratory facilities periodically and reviewed the receipt of samples, laboratory hygiene, sample preparation, assaying method, analysis and data recording. No geophysical tools or spectrometers were used to determine any element concentrations used in this resource estimate.

Criteria	Commentary
	The assaying techniques and QA/QC protocols used are considered appropriate for the data to be used in the Mineral Resource estimate.
Verification of sampling and assaying	The drill hole, sample and assay information was stored in an acQuire database. The collection of data including initial collar coordinates, drillhole designation, logs and assays are controlled to maintain integrity of the database. The data collection and validation process is multi-staged, requiring input from geology technicians, geologists, surveyors and assay laboratories, however the assigned geologist was responsible for the verification of sampling and assaying data for given drillholes or drilling programs. Significant intersections were verified in diamond core by company personnel and typically comprised of quartz veining within moderate to strongly argillic & silica altered host rock. Photographs were taken prior to sampling showing diamond core in original labelled trays with core blocks, metre marks and sample intervals. Remaining half core was retained on site and stored with in the original labelled core trays. Photographs were also taken of washed rock chips from each interval of RC drillholes, the chips were stored in divided plastic boxes labelled with the hole identifier, hole depth was also labelled. Pulps returned from the assaying laboratory are stored on site. Unique sample identifiers were assigned to all samples at the time of sampling and documented in hard copy and digital format before being entered into the geological database. Samples were tracked using a unique dispatch number for each batch of samples sent to the assaying laboratory; any discrepancies identified on receipt of the samples by the assaying laboratory were investigated. Assay reports were checked by the geologist prior to upload into the database and variations from expected values were investigated. Quality control and quality assurance protocols were consistent with industry practice and review of data from initial sampling, assay and reassay values were used for validation. Samples were downgraded in the database and subsequently excluded from the estimate where validation was not satisfactorily resolved. There have been no adjustments to any assay data used in the Pajingo Mineral Resource estimate.
Location of data points	Surface drilling rigs were positioned using surveyed collar pegs, surface holes were located using Real Time Kinetic Differential Global Positioning System (RTK DGPS). Since 2010 conventional surveying methods have been confirmed the accuracy of RTK DGPS locations to within 0.5m laterally and 2m vertically. The drill rig orientation was aligned with front and back sights, pegged out using a sighting compass, an inclinometer was used to align the rig mast with the correct dip angle. Underground drilling collar positions were set out by the mine surveyor using conventional total station method. The rig is aligned with front and back sight positions marked by the surveyor with an inclinometer used to set the correct dip angle. Drilled collar locations and surveyed at the end of each drill program, the surveyed coordinates are tabulated and entered into the geological database. All downhole survey shots were recorded against magnetic north, primary surveys were subsequently converted to local mine grid bearings and both values entered in the geological database. Individual single shot survey records were completed by the driller at 30m intervals, the original records were collated and stored in hard copy for each hole. Single shot survey data was entered manually into the geological database. In addition to single shot surveys, multi shot surveys have been recorded since 1998, the primary record is a digital file that is copied and stored on the Evolution Mining network. Multi shot survey readings were typically recorded at 6m intervals, the extracted digital records were tabulated and entered into the geological database. A local Pajingo mine grid (VN1 Grid) is oriented 37.1 degrees west of magnetic north. Face sample lines were measured from known survey stations to the end of development using a tape measure or electronic distometer. Collar coordinates are determined using the surveyed void position cross referenced to the distance from the known survey station, the vertical position is nominally 1.5m from the floor of the surveyed drive. The topographic surface was based on surveyed points including drillhole collars up to 2012. Underground voids were surveyed using conventional total station surveying methods and cavity monitoring system (CMS) tools. Where voids could not be surveyed, a void shape was created manually based on the design shape and visual inspection of the void. Mined pits were surveyed using total station method.

Criteria	Commentary
	The void model used for the 2014 Mineral Resource estimate was compiled by the site surveyor.
Data spacing and distribution	The estimated lodes were drilled to a nominal 40m x 40m pattern regularly in-filled to 20m x 20m spacing. Face data from ore headings is collected at ~4m intervals depending on the advance of each face. Level separation varies from ~15m to ~30m floor to floor. Sample data was composited downhole to 1m intervals and constrained by the defined lode boundaries. Geological continuity of the Mineral Resource was demonstrated using the existing drillhole distribution and spacing. Geological continuity is further supported by detailed mapping of underground workings. Grade continuity of the Mineral Resource was demonstrated using the existing drillhole distribution and spacing. The mineralised lodes are heterogeneous, grade continuity has been restricted to subdomains determined using the distribution of grade, lode geometry and structural controls.
Orientation of data in relation to geological structure	The main mineralised lodes at Pajingo are generally steeply dipping with an east west strike orientation (VN1 mine grid). Surface drilling has typically been from south to north with an average dip of 60°. Surface drilling is orientated to ensure optimal intersection angle with the reefs. Underground drilling orientation may be limited by available collar locations. Acceptable intersection angles are considered during the drill hole planning process. Many drillholes intersected more than one mineralised lode and were designed to intersect the target lode, occasionally the angle of intersection with adjacent lodes was not optimal. Face sampling is typically taken from exposures perpendicular to the strike of the lode. Low angle and sub parallel intercepts have been excluded from the resource estimate. No orientation bias has been indicated in the drilling data to date.
Sample security	The security of samples was controlled by tracking samples from drill rig to database. Hole identifier and interval depths were marked on diamond core trays and primary RC samples. Photographs were stored showing diamond in original labelled trays with core blocks, metre marks and sample intervals. Remaining half core was retained on site and stored with in the original labelled core trays. Pulps returned from the assaying laboratory were stored on site. Unique sample identifiers were assigned to all samples at the time of sampling and documented in hard copy and digital format before being entered into the geological database. Samples were tracked using a unique dispatch number for each batch of samples sent to the assaying laboratory, allowing the progress if each batch to be monitored. The dispatch number was listed on an assay request form detailing the samples, preparation and assay method required. Any discrepancies identified on receipt of the samples by the assaying laboratory were investigated. Sample tampering or theft has not been an issue.
Audits or reviews	Pajingo drilling data and geological database were reviewed periodically. A review was conducted prior to the acquisition of Pajingo Gold Mine by Conquest Mining in 2010. An internal audit was conducted be Evolution Mining personnel in 2012. An audit of the Resource Estimation process was conducted by Quantitative Group in 2013. A substantial revision of the geological interpretation and estimation methods was prompted by the audit and applied in the 2014 Mineral Resource estimation. Mill to mine reconciliation checks are performed monthly and periodically reviewed for individual lodes.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	Mining and ore processing operations are conducted on ML 1575, ML 10215 and ML 10246. The tenement is owned by NQM Gold 2 Pty Ltd a company wholly owned by Evolution Mining Limited. The area is not subject to any Native Title claims although cultural heritage agreements are in place with the Birriah and Kudjala Peoples. The tenement is in good standing and no known impediments exist.
Exploration done by other parties	The area has been subject to previous soil sampling, RC and diamond drilling, mapping and geophysical exploration by various companies including Battle Mountain, ACM Ltd, Normandy

Criteria	Commentary
	Mining, Newmont, NQM Ltd and Conquest Mining Ltd
Geology	Mining and ore processing operations are conducted on ML 1575, ML 10215 and ML 10246. The tenement is owned by NQM Gold 2 Pty Ltd a company wholly owned by Evolution Mining Limited. The area is not subject to any Native Title claims although cultural heritage agreements are in place with the Birriah and Kudjala Peoples. The tenement is in good standing and no known impediments exist.
Drill hole Information	No exploration has been reported in this release, therefore there is no drill hole information to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to drill hole information relevant to the Mineral Resource estimate can be found in Section 1 – “Sampling techniques” and “Drill sample recovery.”
Data aggregation methods	No exploration has been reported in this release, therefore there are no drill hole intercepts to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to data aggregation methods relevant to the Mineral Resource estimate can be found in Section 1 – “Sampling techniques” and “Drill sample recovery.”
Relationship between mineralisation widths and intercept lengths	No exploration has been reported in this release, therefore there are no relationships between mineralisation widths and intercept lengths to report. This is not relevant to this report on Mineral Resources and Ore Reserves.
Diagrams	No exploration has been reported in this release, therefore no exploration diagrams have been produced. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Balanced reporting	No exploration has been reported in this release, therefore there are no results to report. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Other substantive exploration data	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Further work	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Geological, geotechnical and assay data is collected and stored using an acQuire database. User access to the database is regulated by specific user permission, and validation checks and relational steps are part of the process to ensure data remains valid. Routine validation is undertaken by site personnel during data importation through the use of acQuire workflows to accept or reject data. Further validation checks are performed by site and corporate teams when data is used for interpretation and estimation. Data management is supported by a Database Manager, based in the Evolution office in Perth. Regular back-ups of the database are conducted.
Site visits	The Competent Person is an employee of Evolution Mining Limited and has been a rostered staff member on-site at Pajingo.
Geological interpretation	The level of confidence in the geological interpretation is considered to be good as the geological interpretation is based on drilling and the mapping of underground development. Mineralisation occurs dominantly as amalgams of gold and silver occurring as fine disseminated grains within quartz/carbonate veins. Massive quartz veins and quartz vein breccia are the primary criteria used for defining the shapes of the Pajingo mineralisation body with support from the assay data. Previous interpretations have tended to overstate the thickness and grade of the mineralised veins.

Criteria	Commentary
	Active mining areas (Zed, Sonia, Sonia East, Veracity, Faith, Eva and Olivia) have used an approach refined over the past year to reduce this effect and best represent vein width and grade. Evaluation of these methods via mill reconciliation will continue into 2014 and will be applied to all other areas (Jandam, Vera, Cindy and Janine) as part of the Dec 2014 Resource Statement if performance continues to improve. Alteration and mineralisation of quartz veins has been used to assist identification of the ore zones during the interpretation process. Faults that offset the veins have been mapped in mine development, these are difficult to model from drilling alone. Strike-slip faults with sub-horizontal offsets have been intersected on some levels and are steeply dipping structure sub-parallel to the veins. These faults can cross cut the veins creating barren pillars or overlapping the vein over short strike lengths.
Dimensions	The Mineral Resources at Pajingo comprise a number of vein systems which have varied length, width and dips. Vein width ranges between 0.5 and 12 m and extends around 300 m down dip and along an approximate strike – length of 2,100 m for the various vein systems. Where multiple veins occur generally one vein contains the dominant proportion of metal. Veins have moderate to steep dips (60° - 90°) while width can vary rapidly along strike with down dip width continuity being more consistent. Second order veins 10 to 20 cm wide commonly splay from the main structures and can extend 1 m to 2 m into the footwall or hanging wall.
Estimation and modelling techniques	The Au grade estimation process was estimated by Ordinary Kriging and performed using Vulcan Maptek mining software. Inverse distance weighting interpolation was used to estimate grade into the blocks not informed by Ordinary Kriging. Waste (all material outside the mineralisation domain wireframes) was not estimated due to the highly clustered and limited of data. Therefore, waste was assigned to a below detection limit assay. 1m downhole composites using a minimum 0.3m length and aggregate - merge method, were generated for the drill hole dataset. Intervals with no assays were excluded from the composite data. The following estimation parameters were optimised using Kriging Neighbourhood Analysis (KNA): − Block size − Number of samples − Search range − Block discretization − The following estimation parameters were applied − Octant/quadrant search − Two passes with on average 40%-60% of blocks estimated in the first pass for each domain − Pass one with minimum of 8-12 samples and maximum of 16-24 samples − Pass two with minimum 4-6 samples and maximum of 8-16 samples − Search directions and ranges oriented to the variogram models of each mineralised domain − Search range is applied 2/3 of the sill range − Block discretization of 10 x 5 x 10 (Easting, Northing, Elevation) Top cuts are applied to domains that have extreme values in the grade distribution. The top cut was defined by analysing log probability plots and the mean grade versus the Coefficient of Variance of the mineralised domains. A formal peer review was performed by the internal Evolution resource group and the Pajingo geology manager by comparing the grade and tonnage of current resource estimates to the previous resource estimates. The current estimates take into account new data acquired from grade control and resource definition drilling together with up-dates in the geological interpretation of the deposits through the mapping of new development. Standard model validations have been completed by: − Comparing summary statistics of the data against the estimate − Visualisation between the data against the estimate in sections and plans

Criteria	Commentary
	– Swath plots have been generated on Easting section and plan to check the input composite assays data against grade estimate – Scatter plots between the input composite assays data against grade estimate Silver was estimated using the same estimation parameters as Gold, but is not reported as it not considered material to the operation. The block model size dimension was 7.5m x 5.0m x 7.5m (X, Y, Z). Block size dimension was chosen to reflect the average space of the availability of data. A sub-cell was generated at smaller size 1.5m x 0.5m x 1.5m (X, Y, Z) from the parent block to enable more accurate volume definition against mineralisation domains. The estimation between gold and silver were run independently since the correlation between these two variables is low to moderate. Silver is not considered material to the Pajingo deposits. The resource estimate was constrained by mineralisation domains which were based on the geological and grade continuity. The geological continuity was based on underground development, geological mapping, underground drilling and surface drilling. The grade continuity of the mineralisation domains was cross-checked by boundary analysis method for consistency across the mineralisation domains. Where there is no correlation between the mineralisation domains, a hard boundary between the mineralisation domains was applied to avoid mixing of populations during the resource estimation. Statistical analysis showed the grade distribution is positively skewed with high coefficient of variation due to effect of few high grade outlier samples within the population. The Pajingo resource estimate predominantly used a top cut around the 97th percentile of the grade distribution. Top cuts were determined individually for each domain. A validation of volumetrics between the block model and 3DM wireframes have showed no significant difference. A statistic comparison of the global mean estimated block grades to the mean declustered grade of 1m composite data was performed to ensure the block estimate have globally similar to the composite data. Visual validation of the block model grades and the corresponding drill holes grades were performed on the numbers of sections and plans and show a good correlation and reflect the profile of the mean grades of input data. Swath plots have been generated on sections and plans to check the mean grade of grade estimate against the mean grade of input data and show the grade estimate reflects the input data. A formal peer review was performed internally by Evolution corporate and site.
Moisture	All tonnages are calculated and reported on a dry tonnes basis.
Cut-off parameters	Mineral Resources were reported at a 2.5 g/t Au grade cut-off.
Mining factors or assumptions	Mining methods are assumed to be an equivalent to those currently being undertaken at Pajingo. Pajingo underground deposits have been mined using jumbo development and longhole stoping methods. The key mining production equipment is development jumbos, long hole drills, LHD'S and MT6020 Atlas Copco trucks. All stopes panels have a minimum stoping width of 1.8m, which dependent on ore width is considered as planned dilution. An additional 25% external dilution at a grade of 0.0g/t has been applied to all stopes. This allows for stope overbreak, floor and rill dilution in stope ore extraction. Mining recovery of 95% is estimated for stope panels based on current experience at the underground operation. The current underground infrastructure is suitable to support the mining method with extensions to existing capital development.
Metallurgical factors or assumptions	The ore is to be processed through a traditional CIP/ CIL process plant. The current and estimated future recoveries are 95%. Metallurgical characterisation was conducted in late 2012 on representative samples of current and extensional ore bodies, which confirmed the current assumptions. An operating history of over 20 years supports the metallurgical parameters used.

Criteria	Commentary
Environmental factors or assumptions	Most waste material from underground is utilised as back fill for underground voids. Volumes of waste rock are characterised as non-acid forming (NAF) or, potential acid forming (PAF). Any PAF waste rock in the current mine plan is to be stored in underground voids or within completed open pits. It is the opinion of the site and regulator that all PAF waste will not exceed the volumes as defined in the current mine plan. NAF waste is to be formed into surface waste rock dumps where it is not used as capping material for PAF. During operation NAF waste will be placed as per final landform in the mine closure plan that it does not require further shaping for closure. Current topsoil stockpiles are deemed sufficient for future NAF dumps. Prolonged exposure of PAF materials to aerobic conditions will lead to an increase in acid mine drainage (AMD) which is already demonstrated by water quality within Nancy dam. Potential impacts of AMD are currently localised and managed. At mine closure AMD waters will require remediation. Rehabilitation will include removal of all infrastructure and regulated wastes from site with rehabilitation (flora and terraforming) and ongoing monitoring to take place until rehabilitation success criteria are proven to be robust.
Bulk density	Bulk density was based on a historical bulk density study of the Vera deposit. Subsequent test work during 2012 across all the Pajingo deposits has confirmed the bulk density for ore is 2.65 g/cm^3 and waste 2.54 g/cm^3. Historical bulk density test was measured by dry bulk density where weighed the full core sample in air and water. All the core in selected samples were included in the measurement thus avoiding a bias to the selection of competent samples. The historical bulk density was defined based on the dry bulk density distribution from the samples measurements.
Classification	Measured Mineral Resources are typically supported by drilling data which was mostly less than 20m x 20m spacing, and is additionally confirmed by underground development drives (face samples and geological backs mapping) and infill drilling between underground drives. Indicated Mineral Resources are classified similar as Measured, but with less support from infill drilling and underground data. Typically drill spacing is less than 20m x 20m. Inferred Mineral Resources is classified based on limited data support (no supported from underground data), less confident on the geological continuity, and typically drilling spacing is greater than 20m x 20m. Other aspects that have been taken into account in defining the Mineral Resources classifications are: Data type and Data quality (drill hole orientations; drill hole dh surveys) Statistical performance of the estimate (i.e. slope regression, Kriging Efficiency, number of samples/drill hole used) Geological underground backs mapping The model has been confirmed by successive infill drilling and mine production, which supports the geological interpretation and subsequent classification. The Mineral Resource estimate appropriately reflects the view of the Competent Person.
Audits or reviews	Internal audit and review has been taken during the process Mineral Resource estimation which resulted in no significant issues. The process for geological modelling, estimation parameters and reporting of Mineral Resources is industry standard and has been subject to an independent external review. QG undertook a review of the Mineral Resources during October 2013 and found the adopted process to be an industry recognised technique, applied with sensible parameters. Recommendations by QG have been incorporated in December 2013 Mineral Resource estimate.
Discussion of relative accuracy/ confidence	The relative accuracy of the Mineral Resource estimate is reflected in the reporting of the Mineral Resource as per the guidelines of the 2012 JORC Code. The statement relates to global estimates of tonnes and grade. Interpretation and estimation methods have been refined since the December 2012 statement to better reflect reconciled production through the mill. The estimated relative uncertainty for a

Criteria	Commentary
	Measured resource which is a grade control model equivalent is +/-10%. This is supported by monthly reconciliation figures where ounce variance has averaged <10% since the adoption of the new models in mid-2013. Some areas have performed outside this range and this has led to further refinements which have been applied in this release. It is considered reasonable for an Indicated Resource to have an uncertainty of +/-25%. These figures are reconciled though the mill when mining occurs in these areas, and typically perform within these degrees of confidence. As such, stoping is not recommended until data density is able to increase confidence to the Measured category. In most cases, development above and below in conjunction with 20m x 20m drilling will provide enough data to satisfy these requirements.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	The estimation of Mineral Resources is outlined in Section 3. All Measured and Indicated resource classifications where considered for conversion to Ore Reserves. Mineral Resources are reported inclusive of Ore Reserves.
Site Visits	Regular site visits have been completed throughout the year to gain an understanding of the geological setting, mining methods and mining factors applied to the Ore Reserve.
Study Status	The study to convert Mineral Resources to Ore Reserve is an operational design study. Pajingo is considered to be a mature operation with over 20 years of historical data.
Cut-off parameters	Cut-off grades are not used to estimate Ore Reserve, they are more a generalisation of economic areas. There are numerous cut off values dependent on cost structures applied. A fully costed stoping cut-off grade of 3.3g/t is representative of a mine cut-off grade. All reserves are fully costed within an economic model and based on the proportion of operation and/or capital development required for ore extraction. Thus the cut-off grade varies dependent on these factors, and no one cut-off grade has been used for the Ore Reserve estimation.
Mining factors or assumptions	Pajingo underground deposits have been mined using jumbo development and longhole stoping methods. The mining method adopted is widely termed Modified Avoca whereby stopes are extracted between levels based on geotechnical recommended parameters for stope lengths and heights, then backfilled with loose or consolidated fill before the next retreating stope is extracted. The method has been used extensively at Pajingo throughout its 20 year mine life. Mineable panels have been created based on typical level intervals for the Modified Avoca mining method currently used at the operation. The models have been divided into stope panels 20m long at level interval height (15-20m). No geotechnical evaluation of the reserves has been undertaken, however geotechnical parameters are based on current stoping practices and not expected to diverge greatly from these assumptions. The key mining production equipment is development jumbos, long hole drills, LHD’S and MT6020 Atlas Copco trucks. All stopes panels have a minimum stoping width of 1.8m, which dependent on ore width is considered as planned dilution. An additional 25% external dilution at a grade of 0.0g/t has been applied to all stopes. This allows for stope overbreak, floor and rill dilution in stope ore extraction. Mining recovery of 95% is estimated for stope panels based on current experience at the underground operation. The current underground infrastructure is suitable to support the mining method with extensions to existing capital development.
Metallurgical factors or assumptions	Ore is processed through a traditional carbon-in-leach / carbon-in-pulp process plant at a current rate of around 450ktpa. The current and estimated future recoveries are 95%. An operating history of over 20 years supports the metallurgical parameters used in the Ore Reserve estimation. Metallurgical characterisation was conducted in late 2012 on representative samples of current and extensional ore bodies, which confirmed the current metallurgical parameters. An operating history of over 20 years supports the metallurgical parameters used in the Ore Reserve estimation. Metallurgical characterisation was conducted in late 2012 on representative

Criteria	Commentary
	samples of current and extensional ore bodies, confirmed the current metallurgical assumptions.
Environmental factors or assumptions	Pajingo is current with all environmental approvals and compliant to those conditions set out in such approvals.
Infrastructure	The mine is currently in operation and therefore has adequate infrastructure to support current and future operation.
Costs	Unit costs were derived from FY2014 (July 2013 to February 2014) actual costs. Throughout this period there was some one-off costs due to site restructure that were excluded from the cost estimates. Unit costs were calculated and applied to the economic assessment of the potential Ore Reserves these were; The unit cost estimates included leasing on underground equipment, no other site sustaining costs were used. The performance for the operation for ore tonnage has been lower than expected. The Company is confident issues have been addressed and as such has used an annualised production rate of 350ktpa rather than the eight month performance to date. State royalties are 5%.
Revenue factors	Revenue is calculated using a gold price A$1,350/oz and silver price A$22/oz. No revenue has been created from silver by-products due the uncertain modeling of these grades in the Ore Reserve process.
Market assessment	Gold at spot price.
Economic	The Ore Reserves have been economically evaluated through a standard financial model. All operating and capital costs as well as revenue factors were included in the financial model. This process has demonstrated that the Ore Reserves for the underground operations having a positive NPV.
Social	Currently Evolution Mining has agreements with Traditional Owners and on good terms with neighboring pastoralists.
Classification	Classification of each stope panel is assessed on the proportion of tonnage in each resource classification. If the tonnage in the stope panel is greater than 50% Measured then it is classified as a Proved Ore Reserve. For stope panels greater than 50% Measured and Indicated it is classified as a Probable Ore Reserve. Stope panels that have greater than 50% of Inferred material are excluded from the Ore Reserve estimate. This process leads to some Inferred Resources being included in the Ore Reserve estimate as they will be mined as part of the stope panels. This amount is less than 5% of the Ore Reserve estimate.
Audits or reviews	Internal peer review by Evolution personnel has been conducted in accordance with Evolution’s standards which confirms the stated Ore Reserve and supports the estimation parameters applied. This Ore Reserve has not been audited externally.
Discussion of relative accuracy/ confidence	The accuracy of the estimates within this Ore Reserve are mostly determined by the order of accuracy associated with the Mineral Resource model, the metallurgical input and the long term cost adjustment factors used. In the opinion of the Competent Person, the modifying factors and long term cost assumptions used in the Ore Reserve estimate are reasonable.

Edna May

JORC Code 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

Criteria	Commentary
Sampling techniques	Samples were taken from a combination of RC drilling and Diamond core. From several generations of drilling. Drill spacing was on nominal 25m x 25m spacing with localized areas of 50m x 50m spacing. Holes were vertical or to the South at -60 degrees. The drill hole collar locations were picked up by contract and staff Survey teams using GPS and base station control. Drill samples were logged for lithology, weathering and structure where appropriate. Sampling was carried out over several generations. The majority of drilling (Catalpa Resources) was carried out under Catalpa's quality control and sampling protocols. Historical drilling grades and logging is representative of what has been recorded by Catalpa in later drilling programs. RC samples were collected via cyclone over one or two metre intervals. Samples were riffle split to create a 2kg sample. Wet samples were let to dry and then sampled via riffle splitter. Diamond core is NQ2 diameter. Sampled intervals are matched to geological boundaries and range from 0.25m to 1.2m. The average interval is 1m. From 2008 to 2009 screen fire and leach well methods were utilised. As of 2010 Assaying of half core samples is by 50g fire assay with an AAS finish.
Drilling techniques	Drilling is a combination of 1191 RC and 322 Diamond holes.
Drill sample recovery	Diamond core recoveries have been logged recorded with an average of approximately 95%. RC Drill sample recoveries were not historically recorded. Diamond core is reconstituted into continuous runs for orientation marking and recovery estimations. Core loss (if any) is recorded. Historically RC samples were collected at 1m intervals in individually marked calico bags through a three tier riffle or cone splitter. Sufficient work has not been completed to adequately assess the potential for sample bias.
Logging	Geological logging has been carried out for each drill hole. This includes lithology grainsize, mineralisation, alteration, sulphides and oxidation. RC and Diamond Geological logging has been carried out for each drill hole. This includes lithology grainsize, mineralisation, alteration, sulphides and oxidation. Diamond has also been logged for structural data. Core was photographed. The entire length of RC and Diamond holes was logged and recorded.
Sub-sampling techniques and sample preparation	Core was cut in half and sampled on intervals between 0.2m and 1.2m. RC drilling was completed over several generations. Sampling consisted of three tier riffle splitters or cone splitters. The sample preparation technique for RC and Diamond is considered to be of standard practice within the industry and deemed appropriate. Pre-Catalpa Resources data was utilised on the basis of existing documented historic quality control practices. Later stage drilling follows Catalpa's internal quality control practice which includes a review of laboratory supplied blanks and standards as well as Catalpa supplied blanks and standards. Repeat and duplicate sampling was carried out during the Catalpa generation of drilling. The sample sizes are considered to be appropriate for the lithology and mineralisation style.
Quality of assay data and laboratory tests	Assaying methods used were a combination of fire assay 50g and fire assay 35g dependent on the campaign of drilling. No geophysical tools were used in the compilation of this resource. One standard and blank are inserted every twenty meters. Action was taken on samples returning at greater than two standard deviations. Lab audits were part of the Catalpa policy but no formal record of visits is available.

Criteria	Commentary
Verification of sampling and assaying	Significant intersections historically have been visually verified by staff geologists. Evidence of quarter coring and re-assaying is present for some zones. No record of independent verification exists. One Twin hole and cross cutting holes are present within the data set and appear to highlight no major issues. Primary data was collected by paper logs and transferred to excel in site offices for loading into on-site databases. Some assay techniques were reviewed as they did not liberate all mineralisation within the sample. These samples were statistically reviewed. This affected a minor amount of sample with the majority below the reportable cut off. A review since has shown compensation has had little to no impact.
Location of data points	The collars for the RC and Diamond holes were picked up by ACM, Westonia and Catalpa staff survey personnel. Down hole surveys were completed every 50m by single shot and Eastman camera survey tools. Drilling was conducted using a mine grid rotated 24 degrees clockwise from the national grid system of MGA zone 54. Topographic surface used was digital terrain model (DTM) produced by the companies survey team.
Data spacing and distribution	Drill hole spacing is a nominal 25m x 20m. The drill spacing, spatial distribution and assay type are sufficient to support the classifications applied in accordance with JORC Code 2012 guidelines and is appropriate for the nature and style of mineralisation being reported. Samples have been composited to 2m.
Orientation of data in relation to geological structure	Drilling was angled to provide best opportunity to intercept the mineralisation present as close to perpendicular and true width as possible. No drilling or sampling bias has been noted.
Sample security	Site personnel manage chain of custody. Third party transport company used for transport of samples to lab. At lab, samples are stored in secure area.
Audits or reviews	Audits and reviews have taken place and the resource is an accepted methodology and within acceptable limits

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	Mining Lease M77/88. Owned by Evolution Mining Current operating licenses valid.
Exploration done by other parties	The Edna May Lease was originally explored in 1911. Associated mining and surface exploration continued until 1922 with the cessation of mining. Mining and exploration restarted in 1935 and was completed by 1947. To date mined material was 564,000t@19.6g/t. During this time, the Edna May Reef was mined underground down to 250m below surface. For the period of the Second World War, wolfram and scheelite were mined as by products for the war effort. In 1947 the area had its second hiatus. Exploration in the area re-started in 1984 by ACM. Three main zones were delineated, the wash, pisolitic and Gneiss zones. Shallow RC (RC) drilling was conducted on a 25mx25m pattern. Further drilling down to a depth of 100m was conducted on a 25m x 50m pattern within the oxidised Edna May Gneiss. Minor diamond drilling was also completed. In the 1980's no geophysical techniques were used at Edna May. In 1986 deeper diamond drilling was conducted on a 50m x50m grid to an average of 400m. Two holes of note intersected the Edna May reef system at 500m and 700m depth. Modern exploration has continued along the belt through a combination of classical methodologies

Criteria	Commentary
	including remote sensing and geochemical reconnaissance work. This was often followed up with various drilling techniques including Rotary Air Blast and RC drilling. Prior to Evolution Mining, exploration has been carried out under several different ownerships, ACM, Equinox, Sons of Gwalia, St Barbara, Westonia Mines and finally Catalpa.
Geology	The mineralisation at the Edna May resource comprises Quartz reefs with surrounding low grade halo mineralisation within a package of deformed Gneiss's . See Section 3 for more detail
Drill hole Information	No exploration has been reported in this release, therefore no drill hole information to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to drill hole information relevant to the Mineral Resource estimate can be found in Section 1 – "Sampling techniques" and "Drill sample recovery."
Data aggregation methods	No exploration has been reported in this release, therefore there are no drill hole intercepts to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to data aggregation methods relevant to the Mineral Resource estimate can be found in Section 1 – "Sampling techniques" and "Drill sample recovery."
Relationship between mineralisation widths and intercept lengths	No exploration has been reported in this release, therefore there are no relationships between mineralisation widths and intercept lengths to report. This is not relevant to this report on Mineral Resources and Ore Reserves.
Diagrams	No exploration has been reported in this release, therefore no exploration diagrams have been produced. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Balanced reporting	No exploration has been reported in this release, therefore there are no results to report. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Other substantive exploration data	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Further work	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Paper logs and data were validated prior to entry. Staff Database administrator routinely validates data base for errors. Other checks have been carried out by external consultants. Digital data is interrogated for inconsistencies in data. Any errors found are referenced where possible to the original data or checked in original reports or cross sections.
Site visits	The Competent Person is an employee of Evolution Mining Limited and has been a rostered staff member on-site at Edna May.
Geological interpretation	Mineral Resource estimates were undertaken on the Edna May open pit resource, the Edna May underground resource and the Greenfinch resource. The underground resource was constrained external to the optimized pit shell used to report the open cut resource, while the Greenfinch resource is a standalone estimate to the Edna May resources. Geological interpretation of the Edna May open cut deposit was restricted to the boundary host unit and associated dykes. Mineralisation is associated with high grade quartz reefs that are difficult to interpret and therefore treated as a bulk mineralisation region. The data used was a combination of historical data and more recent drill data. The use of historical drilling data provides a level of uncertainty w.r.t quality control. Due to the bulk mineralisation estimation approach, alternative interpretations are believed to have minimal impact. Mineralisation was estimated within the host rock unit only. For the Edna May underground model the reefs and a halo mineralised envelope was modelled to

Criteria	Commentary
	reflect the more selective mining approach underground mining would provide. The uncertainty involved in the interpretation of the quartz reefs is reflected in the resource classification applied to the underground model Continuity affected by auriferous folded quartz veins, barren intrusive units and cross cutting structures.
Dimensions	The Edna May resources extent Strike = 1100m, Width =90m, Depth = 400m. 500m down dip and open at depth. The Greenfinch resource extent Strike = 900m, Width = 280m, Depth = 240m, 300m down dip and open at depth.
Estimation and modelling techniques	The Edna May open cut block model grades were estimated using multiple indicator kriging (MIK). Extreme grades were cut to a statistical top level. Data was domained into 4 main areas. Software used was Datamine. Search radii x=30m-40m, y=35m-45.5m, z= 10m-13m. The Edna May underground resource was estimated using Ordinary Kriging using the Micormine software package, a flattening technique was used to improve the quality of the estimate. An accumulation technique was used for the reef estimate. The Greenfinch Block model grades were estimated using Ordinary Kriging (OK). Extreme grades were cut to a statistical top level. Data was domained into areas that fitted geologically and statistically. Software used was Micromine. Search radii x=30m-40m, y=35m-45.5m, z= 10m-13m Other estimates have been carried out previously with similar results. No consideration for the recovery of by-products. No consideration has been made for deleterious elements The Edna May open cut block size was 25m x 20m x 10m (x,y,z) sub-blocked to 2.5m x 2.5m x 2.5m (x,y,z). The Edna May underground block size was a regular 5m x 5m by 5m. The Greenfinch block size was 5m x 5m x 2.5m (x,y,z) sub-blocked to 2.5m x 2.5m x 2.5m (x,y,z). Block size reflects mineralisation style and mining method. Bulk open pit and a more selective mining method for the underground. No assumptions were made regarding correlation between variables. Blocks were constrained within the major host rock, reef and halo wireframes. Models of intrusives were used to deplete grade from the estimation. To reduce the influence of high composite grades, cumulative histograms were reviewed and appropriate cuts were applied to the composite grades Block model grades were visually checked against downhole assay grades. Average panel grades were compared directly to with the resource composite grades. Swath plots were generated by easting, northing and RL to compare the input data against the model. The Edna May open cut resource model was also reconciled against previously mined parcels .
Moisture	Tonnages are quoted as dry tonnes.
Cut-off parameters	Cut-off grades were chosen based on geological continuity. The cut-offs used were appropriate for this type of mineralisation, 0.4 g/t Au for the open-cut and 3.0 g/t Au for the underground.
Mining factors or assumptions	Mineralisation modelling reflected the current active mining practices of the deposit. Block size reflected current mining capabilities. Resource reporting used a lower cut-off grade that reflected the current economic cut-off grade of the deposit. See Section 4 for further detail.
Metallurgical factors or assumptions	Metallurgical samples have been taken historically. These included bulk samples of RC and Diamond core. Gold recovery levels were of an expected level for non refractory ore in the industry and well with economic limits. See Section 4 for further detail.
Environmental factors or assumptions	No assumptions have been made within the geological model. Tonnes of primary material are recorded, and used in future waste rock storage planning to reduce potential environmental issues.
Bulk density	Bulk density measurements were taken using the wet method. Frequency of historical measurements is unknown however bulk density estimations used in the resource reflect the actual bulk density in mining the ore body. The results achieved reflect actual bulk density's observed during the mining and milling process. Assumptions were made in allocating bulk density to specific geology, rock types and oxidation

Criteria	Commentary
	state.
Classification	Classification was based on number of samples in the search neighbourhood, minimum number of spatial octants informed, the distance to informing data and geological confidence. All relevant factors have been taken into account. The result is an adequate resource estimate of the deposits.
Audits or reviews	Audits have been completed by third parties to their satisfaction. Internal peer review has also been conducted.
Discussion of relative accuracy/ confidence	The Mineral Resource estimation procedure is considered appropriate. Areas of lower confidence have been classified and flagged appropriately. Where complicated geology exists, a simplified estimation process has been used to minimise inaccuracies in the estimation. The resources can be considered to reflect a global level of confidence. The reported resources for the Edna May and Greenfinch deposits are within an $1800/oz pit shell. The underground resource has been reported from within an optimised stope shell based on a 3 g/t Au cut-off. It was run on material outside the $1800/oz shell. Reconciliations of historical production and the resource model have been performed for the Edna may deposit which is in production.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	The estimation of Mineral Resources is outlined in Section 3. All Measured and Indicated resource classifications where considered for conversion to Ore Reserves. Mineral Resources are reported inclusive of Ore Reserves.
Site Visits	The Competent Person is an employee of Evolution Mining Limited and has been a rostered staff member on-site at Edna May.
Study Status	Edna May has been operating for the past five years and is considered a relatively mature operation. Historic costs and operating parameters have been used in determining the Ore Reserve estimate. Based on the historical data used it is considered the analysis is more accurate than a feasibility study.
Cut-off parameters	The cut-off grade (marginal) used to report the Ore Reserves is derived from the incremental cost of processing ore, additional ore mining costs, metallurgical recoveries, royalties and gold price used during the Whittle optimisation process. A grade of 0.5g/t gold has been used for the Ore Reserve estimate.
Mining factors or assumptions	Steps used to convert the Mineral Resource to Ore Reserve were; pit optimisation, detailed mine design, mine and processing scheduling and financial modelling. Current mining activities at Edna May are undertaken via a conventional drill and blast, truck and excavator open pit operation with 10m high blasting benches mined in three flitches of 3m, 3.5m and 3.5m respectively. The current Edna May pit will be developed in three stages, the initial stage 1 pit and a southern and northern cutback. The optimum pit has been designed following pit slope recommendations by Peter O'Brien and Associates. The E-type grade for a MIK recoverable resource estimate does not require additional mining dilution or mining recovery hence no factor was applied. Minimum mining widths were incorporated in the pit design based on the mining equipment criteria. The Inferred Mineral Resource is used for sensitivity analysis of the optimum pit. The final pit design is based on Measured and Indicated Resource classifications only. External and internal Geotechnical studies are carried out to evaluate the operational designs. Ore Reserves are based on the most recent External recommendations of pit slope berm, batter

Criteria	Commentary
	configuration. The selected mining method does not require additional infrastructure.
Metallurgical factors or assumptions	The Edna May ore is processed through a conventional crush, grind, carbon in leach (CIL) circuit to produce gold doré. In the competent person's view the process for this style of mineralisation is appropriate. The current metallurgical process has been used at Edna May for the past five years. Historically gold recoveries are found to be 92.0%, 93.0% has been used to estimate Ore Reserve as future plant modifications are planned. No assumptions or allowances have been made for deleterious elements.
Environmental factors or assumptions	Additional waste dumping areas are required during the revised life of mine plan. This will require the following approvals: − Mining Proposal for the dump and new diversion drain − Clearing permit for the levy bank − Works approval for construction
Infrastructure	The mine is currently in operation and therefore has adequate infrastructure to support current and future operation.
Costs	Capital costs include process plant modifications, the raising of the integrated waste landform (IWL) for tailings disposal and general sustaining capital. These costs were not included in the optimisation but were included in the evaluation of the project. Operating costs include fixed and variable estimates for reagents, power, consumables, maintenance, labour, administration, mining and accommodation and are based on current third party contracts and historical site data. State royalties are set at 2.5% of revenue while land royalties are calculated using 2.0% of revenue.
Revenue factors	Revenue is calculated using a gold price A$1,350/oz. This price is seen as a representative of current economic forecast for the period.
Market assessment	Gold is sold using a hedged price of A$1,598 per ounce until the end of June 2016 at which time the spot price will be used. Silver credits equate to approximately 1% of total revenue. All silver is sold at spot price. Silver estimates were not included during the optimisation process.
Economic	The project NPV was calculated using capital and operating costs, a gold price of AUD1,350 per ounce and a life of mine (LOM) plan based on the optimised pit reserve. The project NPV was positive and calculated using a discount rate of 8.8%. Sensitivity was conducted on the key input parameters of cost base, head grade and recovery and found to be robust.
Social	Evolution Mining Limited has a close relationship with the community in the nearest town and rural communities. To the best of the Competent Persons knowledge all agreements are in place and are current with all key stakeholders including traditional owner claimants.
Other	Edna May is currently compliant with all legal and regulatory requirements. To the best of the Competent Person's knowledge, there is no reason to assume any government permits and licenses or statutory approvals will not be granted.
Classification	The Ore Reserves are derived from Indicated Resources within the optimum pit design and are classified as Probable Ore Reserves as per usual reporting convention. The Competent Person believes the classification of the Mineral Resource and hence the conversion to Ore Reserve is appropriate.
Audits or reviews	Internal peer review by Evolution personnel has been conducted in accordance with Evolution's standards which confirms the stated Ore Reserve and supports the estimation parameters applied. This Ore Reserve has not been audited externally.
Discussion of relative accuracy/	The accuracy of the estimates within this Ore Reserve are mostly determined by the order of accuracy associated with the Mineral Resource model, the metallurgical input and the long term

Criteria	Commentary
confidence	cost adjustment factors used. In the opinion of the Competent Person, the modifying factors and long term cost assumptions used in the Ore Reserve estimate are reasonable.

Mt Carlton

JORC Code 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td>Sampling techniques</td><td>
RC (RC) and Diamond drilling (DD) methods were used to sample the both V2 and A39 Resource area. The table below summarises drill holes used for the resource estimation. The drill hole data set was trimmed to include sample data relevant to A39 and V2 deposits area.
<table>
<tr><th>Hole Type</th><th>Count</th><th>Total Length (m)</th><th>Average Depth (m)</th><th>Description</th></tr>
<tr><td>RC</td><td>334</td><td>42,215.7</td><td>126.4</td><td>Exploration RC Drilling</td></tr>
<tr><td>DDH</td><td>55</td><td>10,840.6</td><td>197.1</td><td>Diamond Core drilling</td></tr>
<tr><td>RCD</td><td>211</td><td>47,127.8</td><td>223.4</td><td>RC pre-collared diamond core drilling</td></tr>
<tr><td>RC</td><td>9</td><td>530</td><td>59</td><td>A39 GC drill holes</td></tr>
<tr><td>RC</td><td>1092</td><td>24,917</td><td>22.8</td><td>V2 GC Drill holes</td></tr>
<tr><td>Total</td><td>1,701</td><td>125,631.1</td><td>125.7</td><td></td></tr>
</table>
Holes were drilled on 50m centres angled steep to grid south to vertical across most part of the deposit. Areas of significant mineralization were in-filled to 25m by 25m spacing. There are areas where closed spaced drilling has reduced the nominal drill spacing significantly.
</td></tr>
<tr><td></td><td>
Resource definition RC holes were drilled as pre-collars to diamond holes through mainly un-mineralised zones, and then tailed with core through mineralised zones. Grade control RC holes were sampled at 1m intervals by an on-board cone and a riffle splitter. Hole collar locations were set-out and picked up by Evolution Mining Survey Team. Downhole surveys were taken using Camteq Pro-shot for all holes 36m or deeper and 20% of holes shallower than 36m were also downhole surveyed. Grade Control drilling Contractor conducted downhole surveys.
Resource definition hole collars were picked up by Brazier and Motti registered surveyors. Majority of resource definition RC holes were downhole surveyed for dip only, core holes had both azimuth and dip surveyed.
Lithology, alteration and mineralization logging was completed for all holes. In addition density, structural and geotechnical data were collected on oriented core. Core sampling was carried out at a nominal 1m core splits under Conquest Mining QAQC protocols.
</td></tr>
<tr><td></td><td>
Grade control samples taken at 1m intervals were crushed and pulverised at on-site laboratory operated by SGS. However, earlier Grade control samples (Jan-Oct 2011) were prepared and analysed by Genalysis, Townsville. An ICP-4acid digest analytical suite of 10 elements and a fire assay were used by both laboratories using a 50g charge pulp.
HQ/HQ3 core was split in half on site. Samples were taken at 1m intervals and sent to ALS Chemex, Townsville or SGS Townsville Irregular sampling intervals between 0.3 – 1.5m (geologically controlled) were also taken in earlier drilling campaigns.
Aqua Regia/ICPAES analysis for copper and silver (ME-OG46) and 50 gram Fire Assay-AA (AUAA-26) for gold were carried out at ALS, whiles SGS Townsville analysed for gold (FAL505), copper and silver (ICP24R). Core splits and RC chips were dried and prepared to nominal 75 micron for assay.
Preliminary RC sample assays were conducted on four metre composites of one metre sample intervals. On return of composite assays, mineralised intervals were then assayed on a one metre basis. All composites lengths used 50gm charge fire assay technique.
</td></tr>
<tr><td>Drilling techniques</td><td>
Grade Control drilling was conducted using Atlas Copco L8 and Hanjin drills. Face sampling hammers were used to drill 140mm holes. Hole depths ranged 6 to 95m.
Resource definition drilling comprised of RC, RC pre-collars and a diamond tails. RC only holes and pre-collars ranged in depths of up to 270m. 140mm hole diameters were drilled using face sampling hammers. Diamond drilling comprised of HQ and HQ3 core size. Core was oriented using Ezy-Mark and Reflex tools. Diamond holes ranged in depths between 51 and 847m.
</td></tr>
<tr><td>Drill sample</td><td>
RC Grade Control recovery checks were conducted quarterly to confirm nominal recoveries.
</td></tr>
</table>

Criteria	Commentary
recovery	Recoveries were in excess of 85%. Records were kept as part of QAQC monitoring. Diamond core recoveries exceeded 95%. Field recovery records were kept and encoded in the database. Shot core runs were done in bad ground to ensure good core recovery. Overall recovery was very good.
Logging	Geotechnical logging was undertaken by Mining One Consultants Pty Ltd. Rock mass quality data and structural measurements were collected on 12 dedicated oriented diamond core. Structural and laboratory rock strength testing data collected were stored in a database. Data collected on oriented core included; core recovery, RQD, weathering, alteration, estimated rock strength, joint spacing, joint condition, lithological description/units, number of defects, defect type, roughness, infill and infill thickness. RC and diamond core were logged for lithology, alteration, texture, weathering and mineralization. Texture and structure data were recorded for core only. Core was routinely photographed after logging. All drill hole intervals of RC, diamond hole pre-collars and diamond bores were logged in full.
Sub-sampling techniques and sample preparation	Core was cut using a core saw and sampled at nominal one meter intervals from the same side in the tray at all times. Samples were also collected using geological controls at preferential intervals. Core was cut in half through marked orientation lines or on core axis. Quarter core was taken where check samples were required. RC Grade control samples were collected using cone splitters at 1m intervals, whereas resource definition RC holes and pre-collars were sampled by riffle splitters at the same intervals. All samples were collected dry. Core and RC samples were dried and crushed at ALS Chemex, Townsville and SGS Laboratories in Townsville using industry best practice. Samples were pulverized at nominal 85% passing 75microns for assaying. Certified Reference Materials inserted into sample stream covered 5% of sample volume. These included standards, blanks and field duplicates. Initial assays were conducted on four metre composites of initial one metre samples taken during drilling. Significantly mineralised intervals are subsequently re-assayed using 1 metre field re-splits from RC cuttings retained on site. Five pairs of twinned RC and diamond holes within 5m of the original collar locations were drilled for comparison. Equivalent geology and similar assays were established. Sample and grain sizes distribution were consistent and representative of the mineralisation style at Mt Carlton. Shape of mineralised intercepts and thickness, detection limits for poly-metallic assays were also consistent with high sulphidation mineralisation style.
Quality of assay data and laboratory tests	Analytical procedures used by both ALS (Aqua Regia/ICPAES,MEOG46) and SGS (ICP24R) for base metals were partial digestion methods using 2-Acid. Gold was analysed using 50 fire assay charge by both laboratories. ALS - AUAA-26, SGS - FAL505. Nitons were used to measure metal contents as indicative only, it was excluded from the assay database. Size fraction analysis was conducted on all pulverized samples to ensure 85% passing for 75microns standard was achieved. Checks on assay accuracy and umpire analysis was conducted on 4m composites re-assayed from 1m field re-split, 100 sample pulps representative of the mineralization submitted for umpire analysis. Re-assay by ALS of a batch of 271 quarter splits diamond core samples originally assayed by SGS. A gold mean bias of 6% and coefficient of variation of 3 was within acceptable limits. The results indicate no significant relative bias between the two laboratories. Certified Reference Materials (standards, blanks, split duplicates) formed part of the routine internal laboratory QAQC procedure. Accuracy and precision of QAQC data was monitored using control charts. Results indicate good reproducibility and contamination was contained.
Verification of sampling and assaying	Evolutions' General Manager Resource verified ore zones, geological structural exposures in the deposit area and also verified significant intersections in core. This was done as part of the resource review and estimation process. Five pairs of twins were drilled comprising RC and diamond core within 5m of original collar location. A diamond twin pair HC06RC39 and HC06DD004 confirmed geological and mineralogical intersection. Holes formed part of the data used in the estimation process.

Criteria	Commentary
	All drilling and assay data was initially recorded in an Access database, then migrated into Datashed. Data entry has been reviewed by Evolutions Database Administrator with queries routinely addressed by site personnel.
Location of data points	All drillholes within the resource area have been surveyed by licensed surveyor with surveyed coordinates and elevations used in resource estimates.
Data spacing and distribution	Holes drilled on 50m centres towards grid south spaced at 25 north-south sections with a few holes drilled to the north. Mineralisation demonstrates reasonable spatial continuity, high grade domains are controlled by discrete brittle structures and breccia with stylolitic enargite veining proximal to mafic dykes. RC pre collar drilling has been initially sampled as 3-4m composites. Intervals with >0.1 ppm Au were re assayed at 1m intervals.
Orientation of data in relation to geological structure	Dominant mineralised zones trend North East with a component trending North West. Holes were aligned to grid south which is oblique to dominant mineralisation. Structural lineaments data and pit exposures confirm the trend of mineralised domains above. Though, data orientation is oblique to dominant high grade controlling NE trending structure, it is not established whether grid south data orientation has resulted in significant bias.
Sample security	Most samples have been delivered to both Townsville laboratories in person by company personnel with the laboratories forwarding sample submission receipts to check against dispatches. Where samples have been delivered by transport companies, the samples have been stored in locked yards prior to delivery.
Audits or reviews	A number of external audits have been conducted by consultants. Hellman and Schofield Pty Ltd on data quality and sampling techniques. Data quality and assay procedures were found to be suitable.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The current resources are within ML10343, granted to Conquest Mining Pty Ltd for a term of 25 years from the 1st December 2011. The ML area covers 1151.9 ha. Native title agreements are in place for activities within the Mining Lease, and surrounding EPM's. ML 10343 is surrounded by a number of EPM's forming the Mt Carlton project area, with ML10343 within EPM10164.the Mt Carlton project currently covers 875 Km 2, with the EPM's in good standing. A royalty agreement is currently in place between Conquest Mining Pty Ltd and Gold Fields Australasia Pty Ltd.
Exploration done by other parties	Exploration within the Mt Carlton EPM's and ML10343 commenced in the 1970's, with BHP, Ashton Mining, MIM exploration and others exploring the Capsize Range area within the current EPM10164 for porphyry copper and epithermal styles of mineralisation. During the 1990's, MIM discovered the Herbert Creek East prospect, 700m north of the V2 pit. In 2003, Conquest Mining Ltd purchased a number of tenements within the Mt Carlton project area from Xstrata, following the takeover of MIM. In 2006, Conquest Mining discovered the V2 high sulphidation epithermal Au-Cu deposit, and Ag rich A39 deposit, with follow up work within the ML10343 area outlining the current resource.
Geology	The Mt. Carlton high sulphidation deposits at V2 and A39 are located in the Early Permian Lizzie Ck. Volcanic Sequence, which forms part of the Calliope domain in the northern part of the New England Fold Belt. The volcanics were emplaced along the Camboon volcanic arc that formed in the Early Permian. In the project area, a package of andesite lavas and fragmental volcanics overly and underlay a sequence of intrusive and extrusive rhyodacite volcanics, hosting known mineralisation. The volcanic sequence uncomfortably overlies Carboniferous to Permian granite basement.

Criteria	Commentary
	High sulphidation epithermal mineralisation at Mt. Carlton occurred during and in association with rhyolitic volcanism. The rhyolite occurs as a suite of near E-W trending bodies at Mt. Carlton and trend ENE at the V2 and A39 deposit. Along the Capsize trend east of the Mt Carlton deposits, the outcrop expression and off sets of associated strong silica-clay alteration suggests a strong structural control. Later andesitic volcanism overlies the rhyolitic sequences hosting mineralisation. Basaltic to andesitic dykes crosscut mineralisation and mirror pre-existing structures. The dykes are currently interpreted as syn- and post-mineralisation. Mineralisation is hosted within intrusive to extrusive rhyodacite to rhyolite lithologies, consisting of argillic to advanced argillic altered rhyodacitic volcanics and breccias. Within the altered rhyodacite, high sulphidation mineralisation is focused within the upper section of the stratigraphy. The majority of the sulphide veins and sulphide filled breccia /cataclasite zones dip at moderate to steep angles to the shallow dipping host. The overall form of the broader mineralisation halo is mushroom or “wine-glass” in shape, with internal veins and breccias forming discontinuous high grade mineralised zones. Gold mineralisation at V2 is associated with enargite –tennantite copper minerals, with the high grade silver mineralisation at A39 primarily in the form of silver minerals belonging to the argentite group, mainly stromeyerite and acanthite.
Drill hole Information	No exploration has been reported in this release, therefore there is no drill hole information to report. This section is not relevant to this report on Ore Reserves and Mineral Resources. Comments relating to drill hole information relevant to the Mineral Resource estimate can be found in Section 1 – “Sampling techniques”, “Drilling techniques” and “Drill sample recovery”.
Data aggregation methods	No exploration has been reported in this release, therefore there are no drill hole intercepts to report. This section is not relevant to this report on Ore Reserves and Mineral Resources. Comments relating to data aggregation methods relevant to the Mineral Resource estimate can be found in Section 1 – “Sampling techniques”, “Drilling techniques” and “Drill sample recovery”.
Relationship between mineralisation widths and intercept lengths	No exploration has been reported in this release, therefore there are no relationships between mineralisation widths and intercept lengths to report. This section is not relevant to this report on ore Reserves and Mineral Resources.
Diagrams	No exploration has been reported in this release; therefore no exploration diagrams have been produced. This section is not relevant to this report on Ore Reserves and Mineral Resources.
Balanced reporting	No exploration has been reported in this release, therefore there are no results to report. This section is not relevant to this report on Ore Reserves and Mineral Resources.
Other substantive exploration data	No exploration has been reported in this release. This section is not relevant to this report on ore Reserves and Mineral Resources..
Further work	No exploration has been reported in this release. This section is not relevant to this report on ore Reserves and Mineral Resources.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Data is collected and stored using Maxwell’s Datashed™ geological database system. User access to the database is regulated by specific user permissions, and validation checks and relational steps are part of the process to ensure data remains valid. Routine validation is conducted by site personnel during data importation through the use of dedicated import templates with automatic flags for erroneous data. Data management is supported by Evolution’s database specialists who conduct routine validation and historical verification of the data. Regular back-ups of the database is conducted and stored remotely.
Site visits	The Competent Person is a full-time employee of Evolution and undertakes routine visits to the

Criteria	Commentary
	operation.
Geological interpretation	A moderate degree of confidence in the geological interpretation supporting the Mineral Resource estimate is assumed from the interpretation and logging of a substantial drill hole dataset and open-pit mining exposure. The geological model is considered to be simplistic, though captures all significant geological units including barren intrusive dykes. The geological model defining mineralisation at both A39 and V2 is defined by five large domains that correlate to the argillic to advanced argillic altered rhyodacite unit. It is considered that a high risk is present regarding the definition of the mineralisation domains as alternative interpretation suggests that the spatial trend of the high grade mineralisation is considerably more restrictive in spatial context than the diffuse simplistic geological model constrained to the rhyodacite unit defined in the Mineral Resource estimate. The Competent Person deems that the geological model used to define the Mineral Resource estimate is suitable to estimate the global Mineral Resource.
Dimensions	The A39 Deposit forms the western limb of a large planar stratiform bounded gently dipping antiform feature with the V2 Deposit located on the eastern limb of the antiform feature. The strike of the A39 and V2 Deposits is approximately 1,300m by an average width of 300m and 50m in thickness.
Estimation and modelling techniques	The Mt Carlton Mineral Resource was estimated separately for A39 and V2 which have differing metal products. The estimation was using GS3™ software, MP3™ software and Micromine™ software. The principal element estimated at A39 is silver (Ag) with copper (Cu) estimated as a secondary element of significance. The A39 estimate also includes the following deleterious elements which are arsenic (As), sulphur (S), antinomy (Sb), bismuth (Bi), iron (Fe) and zinc (Zn). The principal element estimated at V2 is gold (Au) with Ag and Cu estimated as secondary elements of significance. The V2 estimate also includes the following deleterious elements which are As, S, Sb, Bi, Fe and Zn. The V2 estimate was performed using the Multiple Indicator Kriging (MIK) interpolation method for Au, Ag, Cu and As. Deleterious elements were estimated using the Ordinary Kriging (OK) interpolation method. MIK estimated grades are reported as the E-Type grade. All domain boundaries during the MIK estimate were considered as soft boundaries. The A39 estimate was performed using a combination of OK and Conditional Simulation (CS) interpolation methods. CS estimated grades are reported as the E-Type grade and are based on in-fill grade control (GC) drilling at a 10m by 10m spacing with the broader resource estimated using OK. The block model was generated and estimated in MGA grid Zone 55. Block model (centroids) extents range between 558,305mE to 559,845mE, 7,757,655mN to 7,758,655mN and -127.5mRL to 187.5mRL based on block sizes of 10m by 10m by 5m (X, Y and Z). Block sizes are selected to reflect the selective mining unit (SMU) at Mt Carlton and are smaller than the average drill spacing of 25m by 25m. The Mineral Resource assay database was composited to 2m lengths providing a dataset of 44,585 composites comprising the main elements estimated. The A39 GC assay database was composited to 1m lengths and constituted a dataset of 7,674 composites. A top-cut restriction of 90 g/t Au was applied to the V2 estimate, and 7,000 g/t Ag was applied to the A39 estimate. The top-cuts were applied to limit the influence of extreme grades during the estimate. Block estimation was performed by three search passes for both the MIK and OK interpolations. Search parameters were extended and criteria relaxed accordingly with each subsequent search. The MIK estimation used a first search with a radius of 25m by 25m by 10m (X, Y and Z) which required a minimum of 16 composites and 4 octants and restricted to a maximum of 48 composites. The second pass radius was expanded to 37.5m by 37.5m by 15m (X, Y and Z). The third pass used the same radius as the second pass with a minimum of 8 composites and 2 octants required. The OK estimation is identical to the MIK only varying for the search radius applied. The first search used a search radius of 30m by 30m by 10m with the second and third passes using a search radius of 45m by 45m by 15m.

Criteria	Commentary
	The CS for A39 was superimposed into the Mineral Resource model. The CS block model was constructed from 100 simulations with a node spacing of 2m by 2m by 1.25m. The simulations were generated by a single search pass with a radius of 15m by 15m by 5m. A minimum number of 16 composites and 4 octants restricted to a maximum of 32 composites. The block model (centroids) extents range between 558,645mE to 558,775mE, 7,757,895mN to 7,758,045mN and 66.25mRL to 136.25mRL based on block sizes of 10m by 10m by 2.5m (X, Y and Z). Routine validation of the estimate was completed using grade and tonnage comparisons with previous estimates, swath plots, visual inspection and statistical analysis comparing estimated grades with input composite grades. Review and validation of the estimate was also completed by Evolution personnel. The estimate process is considered appropriate regarding the assumptions implied by the geological understanding at the time of the estimate.
Moisture	Tonnages are estimated on a dry basis. The tonnages of material on stockpiles are quoted on a dry basis.
Cut-off parameters	The cut-off parameter is 0.35g/t Au for V2 and 53g/t Ag for A39 used in the stated company Mineral Resource estimate. Cut-off parameters are based on Evolution's mining (open-pit) and milling costs. The cut-off reflects the current and anticipated mining strategy and practices.
Mining factors or assumptions	The Mineral Resource is further constrained and reported within an A$1,800/oz gold equivalent pit optimisation shell for both A39 and V2. Both V2 and A39 open-pit Mineral Resources are reported after mining depletion using the surveyed surface for the pit as of the 31st December 2013. Current production is by conventional truck and excavator open-pit mining methods with 5m benches taken in two individual 2.5m flitches. Block heights are matched to a panel height of 5m which matches the blasted bench height. Dilution attributed to the difference of the panel to SMU block size is expected to be minimal and within reasonable allowances.
Metallurgical factors or assumptions	The ore is processed through a bulk Sulphide flotation plant. Comprised of the following unit operations; ore reclaim, SAG mill; ball milling; pebble crushing; cyclone classification; bulk flotation and concentrate regrind; concentrate thickening. It is well tested technology used throughout the world for polymetallic orebodies. It has been assumed that deleterious elements will be managed operationally to be blended below the limits set in the Chinese smelter off-take agreements as has been performed since commissioning of the processing plant. The current and estimated future average recoveries at V2 are 89% for Au, 91% for Ag and 91% for Cu. The current and estimated future average recoveries at A39 for Ag is 88% and Cu is 92%. Recent operating history since commissioning supports the metallurgical parameters used in the Mineral Resource and Ore Reserve estimation. Concentrate agreements with Chinese smelters to accept gold and silver concentrate, contain recoverable payment terms based on concentrate grade.
Environmental factors or assumptions	Mt Carlton operates under permitted environmental guidelines with no material concerns defined that will impact the operations viability.
Bulk density	Density values were assigned to the rocks masses depending on their oxidation state with transported material assigned 2.27 tonnes per cubic metre (t/m3), underlying oxidised material assigned 2.35 t/m3, transitional 2.50 t/m3 and fresh material is 2.65 t/m3. Density has been based on the statistical assessment of a dataset consisting of 23,780 measurements collected from Mineral Resource definition drilling prior to the 2009 Definitive Feasibility Study (DFS). Density measurements were attained using the Archimedes principal technique. Density values are deemed appropriate and are supported by operational performance from recent mining activities at A39 and V2.

Criteria	Commentary
Classification	The Mineral Resource is stated inclusive of Ore Reserves and depleted to the mined surface as of 31st December 2013 for both A39 and V2 pits. Mineral Resource classification was originally assigned in the block model based on the number of samples and location of samples used to estimate each block. This is typical of a search pass method being applied. After an internal review of the Mineral Resources the Competent Person deemed it appropriate to reclassify the previously reported Measured material to an Indicated category since the last reported period. The re-classification of the Measured material reflects the JORC 2012 guideline criteria.
Audits or reviews	An external review of the Mineral Resource estimate was undertaken by QG consultants in October 2013. QG recommended that alternative geological interpretations be constructed and reviewed for significance to reduce the geological risk. The external review conducted by QG outlined similar recommendations to improving the Mineral Resource estimate as defined by previous external reviews during the DFS. Recommendations from the QG review have been incorporated into the business development plan for ongoing continuous improvement. All actions recommended by QG are expected to be concluded by the end of 2014.
Discussion of relative accuracy/ confidence	The relative accuracy of the Mineral Resource estimate is reflected in the reporting of the Mineral Resource as per guidelines of the 2012 JORC Code. The statement relates to global estimates of tonnes and grade. Stated Mineral Resources are rounded to 2 significant figures relevant to the accuracy of the estimate. The Mineral Resource estimate was compared with production data for the 2013 calendar period and found to routinely be within a +/- 20% tolerance to the reported milled outcome. The reconciliation outcome supports the reported confidence and subsequent re-classification of material from Measured to Indicated categories for the reported Mineral Resource as of 31st December 2013.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	The Ore Reserve estimate is based on the current Mineral Resource estimate as described in Section 3. The Mineral Resources reported are inclusive of those Mineral Resources modified to produce the Ore Reserve estimate.
Site Visits	The Competent Person is an employee of Evolution Mining Limited and travels on a regular basis to site. Validation of technical and economic assumptions used in the preparation of this Ore Reserve estimate occurs during these site visits.
Study Status	Mt Carlton is considered to be a young operation with 1.5 year of historical data. Performance of the current & on-going mining activities has demonstrated that mine plans are technically achievable and economically viable considering the modifying factors. On this basis the analysis is considered at a higher level than a Feasibility Study.
Cut-off parameters	Two cut-off grades have been calculated based on the current and forecasted costs and modifying factors, forecast over a period greater than 3 years. These cut-off values are; Fully Costed – cut-off includes all operating costs associated with the extraction, processing, transport and smelting of ore material; Incremental – cut-off grade applies to material that will be mined in the process of gaining access to economic material. Ore Reserve are reported above the following cut-offs; − Gold in the V2 pit at 0.90g/t

Criteria	Commentary
	– Silver in the A39 pit at 53g/t
Mining factors or assumptions	The methodology used to convert the Mineral Resource to Ore Reserve can be described as optimisation of existing open pit operations through standard mine planning process steps of pit optimisation, mine design, mine schedule and financial modelling. Factors and assumptions have been formed from existing operating technical assumptions and cost models. On this basis the analysis is considered at a higher than Feasibility Study. Current mining at Mt Carlton open pit is undertaken via conventional truck and excavator fleet to extract ore material to the ROM, waste material to the waste rock dumps and stockpiling and reclaim of lower grade material. The current mining activities show the appropriateness of this mining method as the basis of the Ore Reserve. Ore dilution and recovery loss is specifically accounted for in the Mineral Resource modelling method and no additional mining dilution or recovery factors are applied to the Mt Carlton Pit Ore Reserve estimate. This assumption is supported by the actual reconciliation between resource model and mill performance at Mt Carlton to date being within acceptable uncertainty range for the style of mineralisation under consideration. External and internal Geotechnical studies are carried out to evaluate the operational designs. Ore Reserves are based on the 2009 Geotechnical Report from Mining One recommendations of pit slope berm, batter configuration. Inferred material is excluded from the Ore Reserve and treated as waste material, which incurs a mining cost but is not processed and hence does not generate any revenue. The optimisation evaluation shows the ultimate pit size is insensitive to Inferred Resources. The selected mining method does not require additional infrastructure
Metallurgical factors or assumptions	The ore is processed through a bulk Sulphide flotation plant. Comprised of the following unit operations; ore reclaim, SAG mill; Ball milling; pebble crushing; cyclone classification; bulk floatation and concentrate regrind; concentrate thickening. It is well tested technology used throughout the world for polymetallic orebodies. It has been assumed that deleterious elements will be managed operationally to be blended below the limits set in the Chinese smelter off take agreements as has been performed since commissioning of the processing plant. The current and estimated future average recoveries at V2 are 89% for Au, 91% for Ag and 91% for Cu. The current and estimated future average recoveries at A39 for Ag is 88% and Cu is 92%. Recent operating history since commissioning supports the metallurgical parameters used in the Ore Reserve estimation. Concentrate agreements with Chinese smelters to accept Gold and Silver concentrate, contain recoverable payment terms based on concentrate grade. The Ore Reserve has been estimated that the concentrate will deliver above the specification payable grades over the life of the mine.
Environmental factors or assumptions	Mt Carlton is current with all environmental approvals and compliant to those conditions set out in such approvals. Environmental rehabilitation plans are produced and cost of the mine closure rehabilitation work is accounted for in the financial evaluation model.
Infrastructure	The mine is currently in operation and therefore has adequate infrastructure to support current and future operation.
Costs	Capital and operating costs have been determined based on the current operating cost base modified for changing activity levels and reasonable cost base reductions over the life of the mine. On this basis the analysis is considered at a higher level than a Feasibility Study. Site unit costs are applied both as break even site cost used to determine ultimate pit shell and marginal site cost used to define ore waste cut off boundary within the ultimate pit shell. The break even cost base is predicated on similar levels of site activity to recent history with planned cost improvements built in. No cost impact is expected from deleterious elements and no costs have been included in the Ore Reserve estimate for these. Transport costs have been built up from first principles consistent with the application and input assumptions for these costs used by the current operation. State Royalties - 5%; Third party royalty – 2.5%

Criteria	Commentary
Revenue factors	Revenue is calculated using a gold price A$1,350/oz, silver price A$22/oz and a copper price A$3.00/lb. A typical 3 year trailing average has not been used to set the commodity pricing. Instead a position has been set based on mean broker estimates and the Company's longer term view of these commodities.
Market assessment	Gold and silver concentrate is sold to Chinese smelters under commercial agreements, these agreements are for life of mine terms.
Economic	To demonstrate the Ore Reserve as economic it has been evaluated through a standard financial model. All operating and capital costs as well as revenue factors were included in the financial model. This process has demonstrated that the Ore Reserves for the Mt Carlton open pit has a positive NPV. Sensitivity was conducted on the key input parameters of cost base, head grade and recovery and found to be robust.
Social	Currently Evolution Mining has agreements with Traditional Owners and on good terms with neighbouring pastoralists.
Other	There are typical risks for an open pit operation such as heavy rain fall events and geotechnical risks. These risks are managed through the implementation of various risk management mechanisms as far as practical.
Classification	The Ore Reserves are predominantly derived from Indicated Resources. This classification is based on the density of drilling, the orebody experience and the mining method employed. The only Probable Reserves derived from Measured Resources are those reported in known and quantified stockpiles. It is the Competent Person's view that the classifications used for the Ore Reserves are appropriate.
Audits or reviews	Internal peer review by Evolution personnel has been conducted in accordance with Evolution's standards which confirms the stated Ore Reserve and supports the estimation parameters applied. This Ore Reserve has not been audited externally.
Discussion of relative accuracy/ confidence	The accuracy of the estimates within this Ore Reserve are mostly determined by the order of accuracy associated with the Mineral Resource model, the metallurgical input and the long term cost adjustment factors used. In the opinion of the Competent Person, the modifying factors and long term cost assumptions used in the Ore Reserve estimate are reasonable.

Mt Rawdon

JORC Code 2012 Edition – Table 1

Section 1 Sampling Techniques and Data

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td rowspan="2">Sampling techniques</td><td>
The deposit has been sampled using Percussion (PERC), RC (RC) and diamond drill holes (DD) on a nominal 25m x 25m grid spacing from surface (256mRL max) down to 0mRL, and nominal 50m x 50m below this RL. FY14 drilling focused on tightening up most of the -0mRL orebody to 25m x 25m spacing.
Holes were generally drilled on grid northings and angled towards grid west or east.
<table>
<tr><th>Type</th><th>No. of holes</th><th>Total metres</th><th>% of Total metres</th></tr>
<tr><td>Percussion</td><td>45</td><td>5,997</td><td>6.9%</td></tr>
<tr><td>RC</td><td>246</td><td>17,925</td><td>20.7%</td></tr>
<tr><td>DD</td><td>142</td><td>24,942</td><td>28.7%</td></tr>
<tr><td>RC/DD</td><td>96</td><td>37,916</td><td>43.7%</td></tr>
<tr><td>Totals</td><td>529</td><td>86,780</td><td>100.0%</td></tr>
</table>
Ave. hole length = 164m
</td></tr>
<tr><td>
All holes drilled since 1979 have had collar locations picked up by licensed surveyors. The holes were downhole surveyed on arbitrary 50m intervals either using Eastman Single Shot down-hole camera (historic drilling) or, more recently, a Reflex Electronic single shot camera to monitor changes in inclination and declination. The RC samples were collected by cone or riffle splitter to ensure sample representivity. Diamond core was used to obtain high quality samples that were logged for lithological, structural, geotechnical, density and other attributes. Blank samples and certified standard samples are routinely inserted during drill sampling operations as per standard QAQC practices.
Historic diamond drilling (HQ or NQ2) was sampled on 2m or (more recently) 1m intervals, cut and sampled to yield sample weights under 7kg. Since November 2011 all diamond core has been NQ2 size, sampled on geological intervals (0.5 to 1.5m). Prior to November 2011 RC samples were taken at 1m intervals and composited to produce a 2m sample for dispatch to an independent laboratory for analysis. The RC samples submitted are not routinely weighed, however tend to be around the 3 – 5kg weight.
</td></tr>
<tr><td>Drilling techniques</td><td>Since Nov 2011 all holes have been drilled as 100% RC or NQ diamond with an RC precollar. RC precollar depths range up to 264m but are normally around 200 – 220m. Hole depths for the database used in the model, range from 48m to 250m and averaging 133m (Percussion), 20m to 314m and averaging 73m (RC), 40m to 471m and averaging 176m (DD) and 240m to 930m averaging 395m (RC/DD). Historical drilling between 1979 and 1984 comprised Open Hole Percussion (Rotary Air Blast drilling) and only limited diamond core drilling. There is doubt regarding the quality of the early rotary drilling however these holes only form a small component of the database and are in the upper reaches of the orebody long since mined away. From 1984 to 1998 Conventional Reverse Drilling was used, with RC hole diameters ranging from 5.25 inches (134mm) to 5.75 inches (146mm) in diameter. After 1998 the Face Sampling RC system has been utilised, allowing for sample return directly from the bit face through the inside of the drill string thus greatly reducing the risk of potential down-hole contamination. Recent core from infill holes have not been oriented, however many of the drill holes from earlier years were oriented by the spear method.</td></tr>
<tr><td>Drill sample recovery</td><td>During the 1995 Feasibility Study, a study of all core available from the deposit to that time (10,801 data points) determined the average core recovery to be 98.3%. An Oct 2000 independent database validation report by RSG for Equigold NL notes ‘similar high levels of RC sample recovery were also noted, with very few wet samples’, as has been the subjectively observed experience with Mt Rawdon drilling in recent years. In the FY12 geotechnical drill hole program where 8 holes were drilled for 3,110m, core recovery averaged 99%.No current data for RC recovery is being collected however this will be introduced in the FY15 drilling program, to confirm the very high recovery rates that have been experienced historically at MRO.</td></tr>
</table>

Criteria	Commentary
	Drill core is routinely meter-marked-up with depths checked against the depth recorded on the core blocks and rod counts are routinely carried out by the drillers. Any discrepancies are quickly resolved in conference with the drillers. The drilling contractors (AED - Townsville) have been drilling at Mt Rawdon for over six years and thus have gained valuable knowledge of the rocks drilling characteristics and have honed their standard procedures to maximize recovery and meter advance rates. Placer's 1995 'Mt Rawdon Recovery and RQD Study' (10,801 data points) explored this possibility thoroughly and concluded 'there is no direct connection between recovery/RQD and gold grade.
Logging	All holes in the drillhole database have been geologically logged. Many early holes (pre-2001) and a small proportion of recent holes have also been geotechnically logged. Drilling completed post-1984 up to November 2011 (Placer Pacific Ltd and Equigold NL) was audited by RSG and found to be of suitable quality for resource determinations, and that appropriate diligence had been exercised. Several geotechnical programs have been implemented for pit wall design since the project began, the most recent ("Geotechnical Assessment Extended Open Pit Mining" by Peter O'Brian and Associates in July 2013) based on the drilling and intensive geotechnical logging of 8 HQ/NQ diamond holes for 3,110m in 2012. Recent infill drilling is not orientated or geotechnically logged. Logging of diamond core and RC chips recorded lithology, oxidation, colour and mineralisation. Core was photographed in both dry and wet form for geotechnical holes, and wet form for infill holes. All drillholes were logged in full.
Sub-sampling techniques and sample preparation	All diamond drilling is HQ or NQ2 size. One recent campaign of infill holes (FY13) was whole-core assayed however all other drillholes have been cut in in half (NQ2) or quarter (HQ) onsite using a manual core saw for sample submission. RC samples were collected on the rig using cone or riffle splitters. Due to the tight nature of the volcanic rock at Mt Rawdon all RC samples were dry when collected (although no data exists to confirm this for historic holes) The sample preparation of diamond core follows industry best practice, involving oven drying for 12hrs at 105°C, coarse crushing down to ~5mm followed by riffle splitting (if sample size is greater than 3.5kg). Each sample is then entirely pulverised for up to 7 minutes in an Essa LM5 pulveriser to >80% less than 75 micron. Approximately 250g of the pulverised product is scooped from the LM5 bowl into labelled pulp envelopes and placed into cardboard boxes for easy handling. The LM5 bowl is vacuumed between samples and at the completion of a batch each bowl is 'washed' using a coarse barren quartz wash. Field duplicates of the RC samples were taken at regular intervals (one field duplicate per 30m) using a spear sampling method. As the sampling method for the primary and duplicate is different, it could be argued that they are not true field duplicates. The performance of the field duplicates is considered poor but some of the error may be attributable to the sampling method. Field duplicates were taken using a riffle splitter and cone splitter for historic RC holes or spear sampling of the sample pile for the recent RC drilling programs. The sample sizes are considered appropriate to correctly represent the disseminated low-grade Mt Rawdon orebody.
Quality of assay data and laboratory tests	The detection limit for the analyses is 0.01ppm Au and 1.0ppm Ag. The samples were dried, crushed and pulverised to produce a 50g charge for gold analysis by fire assay (method FAA505) followed by an AAS finish at either the SGS Townsville or ALS Brisbane laboratories. Silver analyses were obtained by methods DIG21R and AAS21R (i.e. a 3-acid digest followed by AAS analysis). Due to the low-silica nature of the mineralisation the 3 acid digest is considered adequate for total digestion. Not applicable as this type of tool/instrument is not used at MRO. Standard field QC procedures employed involve the use of certified reference material as assay standards, along with blanks and field duplicates. Since November 2011 when Evolution Mining took control of Mt Rawdon Operations there have been 37,668 gold assays along with 1785 field duplicates (4.7%), 2028 standards (5.4%) and 548 blanks (1.5%). The total QC insertion rate is 11.6% (~1:9). Acceptable levels of accuracy and precision have been established.
Verification of	As there is no visual control to the low grade Mount Rawdon ore-body this has not been deemed

Criteria	Commentary
sampling and assaying	necessary. No twinned diamond holes are known to exist in the historic drilling data however recent hole MTRRCD0023 had three wedges drilled off it through a rich section of ore, mimicking a twinned hole. The very high grades in the initial hole were not duplicated in the wedge holes, indicating the very localised nature of very high grades in the deposit. Primary logging data was collected by the Mine Geologists on paper hardcopy templates using the standard site legend then manually entered into Excel templates and saved as .csv files. The information was first validated by the Geology Manager then uploaded into a SQL database server. No adjustments or calibrations were made to any assay data used in this estimate.
Location of data points	Historically all RC and Diamond drill holes at the Mount Rawdon Project have the collar positions accurately surveyed by a licensed surveyor. Inclined drill holes had down-hole survey readings taken at arbitrary 50 meter intervals, to monitor changes in inclination and declination. The historic drilling utilised an Eastman Single shot down-hole camera. Recent drilling from 2007 onwards has used the more advanced Reflex Electronic single shot camera. In the most recent drilling campaigns surveys were taken at 30m intervals. Hole collars have been routinely surveyed in accurately by the mines survey team. The grid system used is AGD84. Local RL is the same as regional RL, in that no increment has been added to the local RLs. Excellent topographic control has been achieved by the mine survey team on a routine basis inside the open pit.
Data spacing and distribution	The data spacing nominally 25 x 25m above 0mRL and either 25 x 25m or 50 x 50m below 0mRL. The ongoing Resource Definition drilling program is planned to infill the whole resource to 25x25m spacing eventually. The mineralised domains used have demonstrated sufficient continuity in both geological and grade continuity to support the definition of Mineral Resource and Reserves, and the classifications applied under the 2012 JORC Code. Samples were composited to 6m, this being the deemed the most appropriate length after studies by AMC Consultants in 2011 and the Perth MRG in 2012.
Orientation of data in relation to geological structure	As Mount Rawdon is a disseminated sulphide low-grade gold deposit with no prominent key mineralisation orientations it is believed that the orientation of sampling has not caused biased data. As above, no sampling bias is considered to exist in the data used to inform this block model.
Sample security	Chain of custody is managed by Evolution Mining. Samples are stored on site and collected by the local transport company (McKay's Transport) for next-day delivery to the TNT depot in Bundaberg and from thence truck delivery to either the Townsville (SGS) or Brisbane (ALS) laboratories. Whilst in storage they are kept in a locked yard. Sample batches are receipted
Audits or reviews	The site geological database at Mt Rawdon is an SQL based relationship database supplied by acQuire Technology Solutions Pty Ltd. Data was migrated to the acQuire system from the previous Datashed site database in 2010. As part of the current Mineral Resource estimate, Evolution has undergone a series of validation processes to ensure that the historic data now contained in acQuire reflects the data previously captured in the historic database and the database is considered to be of sufficient quality to carry out resource estimation. A similar decision was reached by AMC Consultants during a previous model run in June 2012. No laboratory audits or visits have been carried out in recent years of the SGS Townsville or ALS Brisbane lab facilities.

Section 2 Reporting of Exploration Results

Criteria	Commentary
Mineral tenement and land tenure status	The mining and processing operations occur on nine granted MLs – ML1192, ML1203, ML1204, ML1206, ML1210, ML1231, ML1259, ML50119 & ML80095 all situated in the North Burnett Shire some 15km south east of Mount Perry township. They are all currently held by LGL Mount Rawdon Operations PTY LTD, a subsidiary of Newcrest Mining Ltd. They are only weeks away from being lodged for transfer across to Mount Rawdon Operations, a subsidiary of Evolution Mining Limited. Mt Rawdon is a mature operation having been in continuous production since 2001 and all of the mine leases are held in good stead, with sufficient length of tenure remaining to completely mine and process the known orebody. No native title issues or historical sites exist on the leases. The mine and processing plant operate under an environmental agreement with the Queensland state government. A royalty is paid to the Queensland state government based on gold ounces produced.
Exploration done by other parties	The Mt Rawdon deposit has been explored by a large number of companies since its discovery by Noranda in 1969. These have included Newmont, Getty Oil, Resolute and BHP and most notably Placer since 1985, who produced the first feasibility study. Equigold NL acquired the project in 1998, completed a new feasibility study in 1999 and started mining and processing the deposit in 2001.
Geology	Mt. Rawdon is an IRGS (Intrusive-Related Gold System) low grade gold-silver deposit occurring in rhyodacitic rocks forming the basal sequence of the Aranbanga Volcanic Group of mid- to late Triassic age. The principal host rocks are a thick pile of volcaniclastics and various dacitic and rhyodacitic plugs and sills which intrude it. The area is also cut by a sequence of younger post-mineralisation dykes and sills ranging from andesite through to rhyolite composition, all of which lack economic gold values. The volcaniclastics are typically massive, clast-supported and poorly sorted with bedding rarely observed. Mineralising hydrothermal fluids passing through the volcanic pile via natural porosity and micro-fractures have led to a wide range of alteration styles and intensity, notably sericitic and albitic. Economic mineralisation is closely associated with finely disseminated sulphides (principally pyrite) infusing the host rocks as well as discrete irregular pyrite-base metal patches and veinlets. Gold grade generally increases with an increase in sulphide content. The overall orebody takes the form of a thick tabular body dipping moderately to the west and plunging to the south-west. Up to 15% of recovered gold is extracted by the gravity circuit (Knelson Concentrators) and the low work index of the altered host rocks (particularly the rhyodacite plug) assists in reducing processing costs.
Drill hole Information	No exploration has been reported in this release, therefore no drill hole information to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to drill hole information relevant to the Mineral Resource estimate can be found in Section 1 – “Sampling techniques” and “Drill sample recovery.”
Data aggregation methods	No exploration has been reported in this release, therefore there are no drill hole intercepts to report. This section is not relevant to this report on Mineral Resources and Ore Reserves. Comments relating to data aggregation methods relevant to the Mineral Resource estimate can be found in Section 1 – “Sampling techniques” and “Drill sample recovery.”
Relationship between mineralisation widths and intercept lengths	No exploration has been reported in this release, therefore there are no relationships between mineralisation widths and intercept lengths to report. This is not relevant to this report on Mineral Resources and Ore Reserves.
Diagrams	No exploration has been reported in this release, therefore no exploration diagrams have been produced. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Balanced reporting	No exploration has been reported in this release, therefore there are no results to report. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Other substantive exploration data	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.
Further work	No exploration results have been reported in this release. This section is not relevant to this report on Mineral Resources and Ore Reserves.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	Commentary
Database integrity	Data is collected and stored using an acQuire software database system. User access to the database is regulated by specific user permissions, and validation checks and relational steps are part of the process to ensure data remains valid. Routine validation is conducted by site personnel during data importation through the use of acQuire workflows processes to accept or reject data. Data management is supported by Evolution personnel (database technicians) based in Perth with routine validation and historical verification of data being performed. Regular back-ups of the database are conducted. A complete validation and checks of data was performed by the Competent Person during the course of the estimate to verify and correct inconsistencies and errors in the data extracts from acQuire.
Site visits	The Competent Person is an employee of Evolution Mining Limited and has been a rostered staff member on-site at Mount Rawdon..
Geological interpretation	The confidence of the geological interpretation is considered to be reasonably high. Confidence is supported and based on geological knowledge acquired from open-pit production data, detailed drill hole logging, assays and pit mapping. The interpreted geological model consists of a single Au mineralisation event; comprising fine disseminated to stringer hosted sulphide (pyrite) veinlets hosted within a sequence of dacite intrusives and dacite-rich volcaniclastics. The mineralisation is not constrained to a single lithology, though it is intruded by a later stage sequence of barren intrusive dykes. Statistical analysis denotes a mixed (bimodal) population of Au grades due to the spatial integration of the sulphide mineralisation. Mineralisation is modelled and constrained as a separate domain by creating a grade envelope relating to a 0.1 gram per tonne (g/t) Au cut-off based on non-composited assays. Geological Interpretations were modelled in both Surpac™ Software and Leapfrog™ Software by sectional methods on a +/-25m window slice throughout the deposit. Mineralisation and lithological outlines were terminated at half the drill hole spacing beyond the last known section. The grade continuity of the mineralisation was cross-checked by contact analysis methods for consistency across the dacite intrusives and dacite-rich volcaniclastic contacts. The consistent grade distribution across the lithological units confirms the grade interpretation method used to delineate mineralisation is adequate. As additional data is collected and collated, the geological interpretation is continually being updated to refine the spatial extents of the geological wireframes. The input data has been validated by geological personnel at Mt Rawdon and is considered acceptable for determining the geological model. The geological interpretation is considered robust and no alternative interpretations are proposed due to the confidence in the interpretation verified by mining.
Dimensions	The Mineral Resource area encompasses the mineralisation domain (i.e. modelled 0.1 g/t Au envelope) which has a moderate southerly plunging ovoid shape with approximate dimensions of 950m (north) by 550m (east) and is distinguished between the 260mRL and -400mRL (or 650m in elevation).
Estimation and modelling techniques	The Mt Rawdon estimate is reported for Au (primary economic metal of interest). The Au grade for the estimate is in parts per million (ppm). Secondary (by-product) and deleterious elements of interest also estimated included Ag (ppm), C (%) and S (%). These elements are used as indicative grades for a mining parcel to monitor and ensure processing performance and scheduling. Ag is a by-product to the extraction of Au, and is incorporated in the interpolation of the final pit design, though due to the low grade tenor is deemed to not have a material effect on the outcome. The estimate of C and S is used to identify Potential Acid Forming (PAF) material within the model. The Au estimation was completed using Multiple Indicator Kriging (MIK) a non-linear interpolation

Criteria	Commentary
	technique and reported as the E-Type grade. The MIK technique was selected as a suitable method to account for the bimodal, strongly skewed grade distribution with a high co-efficient of variation (CV) not amenable to linear estimation techniques. The Ag estimate was performed using Ordinary Kriging (OK) which is a linear estimate technique and deemed suitable to estimate Ag which had a low CV. The C and S estimate was performed using Inverse Distance Cubed (ID3). The dataset for both C and S assays is only a partial representation (26% and 37% respectively) of the total Au assay dataset. ID3 is deemed appropriate for the estimate of C and S due to the distribution and partial data present. Respectively the confidence in the C and S estimates is low. The estimates were all performed using Isatis™ Software with post-model editing and validation performed using Micromine™ Software. The three dimensional model (3DM) and digital terrain model (DTM) wireframes for the estimated domains, incorporating lithological, mineralisation, oxidisation and topographic files were used to constrain the resource estimate. Blocks from the block model were coded based on these volumes/surfaces if in/out of 3DM or above/below a DTM surface by block percentages >50%. All surfaces were treated as hard boundaries in the estimate. The estimated domains identified in the model are the mineralized domains denoted as either above/below the Quartz-Feldspathic-Biotite Porphyry (QFBP) (a late stage barren intrusive) within the modelled 0.1 g/t grade envelope. Waste domains were not estimated due to the highly clustered and minimal quantity of data available. Waste domains were assigned the median statistical grade of the limited composite dataset in each case. 6m down hole composites (using a minimum 0.5m length and aggregated merge method) for the entire drill hole dataset was generated. Each composite was assigned a corresponding domain by using a selection variable generated from the interpreted wireframes. Exploratory data analysis (EDA) was undertaken on the composite data for each domain and element. Declustered statistics were also reviewed and incorporated into the interpolation. The composite domains representing mineralisation was divided into 14 indicator thresholds. The statistics for each indicator threshold were reviewed during the interpolation process. Spatial continuity/variography modelling was completed for each indicator threshold. Directions of continuity were similar to the interpreted control on mineralisation with varying degrees of anisotropy for the below QFBP domain. The above QFBP mineralized domain was modelled was modelled omnidirectional as no distinct anisotropy was observed. Block model extents range between 374,610mE to 375,990mE, 7,203,410mN to 7,204,690mN and -445mRl to 285mRl, based on a regular block model with block size dimensions of 20m by 20m by 15m (X, Y, & Z). Block size dimensions were chosen to reflect the Selective Mining Unit (SMU) height and accommodate drill spacing and sample support criteria. Estimate parameters were optimised using Quantitative kriging neighbourhood analysis (QKNA). Parameters optimised include: − Search parameters − Number of samples (minimum and maximum) − Block discretisation Key Au and Ag estimate parameters applied include: − Quadrant search − Two passes with >80% blocks estimated in first pass for each domain − Pass 1 with minimum of 8 samples and maximum of 24 samples − Pass 2 with minimum of 6 samples and maximum of 12 samples − Search directions and ranges orientated to variography and mineralisation domain trend − Block discretisation of 3 x 3 x 2 (X, Y, & Z) − Ag estimates applied a cut-off threshold of 14 g/t Ag and 8g/t Ag after 10m Search criteria were relaxed in the second pass to limit conditional bias concerns and accommodate wider spaced data. No top-cut was applied to the Au estimate, but instead the influence of extreme grades was controlled by the MIK technique.

Criteria	Commentary
	Final model block grades are the weighted average accumulation based of the domain proportion of the block. MIK estimates for Au have been checked against alternate OK and ID3 estimates (utilising the same data) and also reconciled with previous estimates. Correlation between elements is considered to be poor and the estimate of Ag, C and S is not amenable to co-kriging options. A comparison between the mean grades from the drill hole composite data and the block estimates was performed to ensure they were similar and the estimate unbiased in a global sense. Change of support checks of the input grades to the composites and block model identified no issues with the MIK estimate. Grade - tonnage curves were generated to review cut-off grade sensitivity. This was also compared with previous estimates. Standard model validation has been completed using visual (i.e. input composite/raw drill hole data) and numerical methods. Sufficient spot checks have been carried out on a number of block estimates on sections and plans. Swath plots have been generated on sections (Easting, Northing and Elevation) to check the input composited assay data against block grade estimates and found no issues. A formal peer review was performed by Evolution personnel external to the estimation process.
Moisture	Tonnages are estimated on a dry basis. The tonnages of material on resource stockpiles are quoted on a dry basis.
Cut-off parameters	The cut-off parameter is 0.233 g/t Au for the stated Mineral Resource estimate. Cut-off parameters are based on Evolution's mining (open-pit) and milling costs. The cut-off reflects the current and anticipated mining strategy and practices.
Mining factors or assumptions	The Mineral Resource is further constrained and reported within a $1,800 pit optimisation shell (Maximum depth approximately equivalent to -210mRL). Current production is by conventional truck and excavator open-pit mining methods with 15m benches. Block heights in the model are matched to the 15m mining bench heights (determined to be the SMU height). See Section 4 for more detail.
Metallurgical factors or assumptions	Processing is by conventional gravity concentration and cyanide leaching techniques with an achieved average recovery of 91% (based on historic performance) for gold.
Environmental factors or assumptions	PAF material is a consideration for the Mt Rawdon operation due to the style and nature of the mineralisation. PAF material is managed in accordance with all regulatory requirements as part of the operational procedure and is not considered to be of material importance. PAF material is calculated to be <10% of the total waste tonnage to be extracted within the current final pit design. No other significant environmental issues have been encountered or are anticipated.
Bulk density	Density measurements have been collected using the Archimedes Method. Density data has been collected over the operations life, comprising a dataset in excess of >6,500 samples. All care has been taken to exclude any erroneous data during collection. Density values were assigned in the Mineral Resource estimate based on the lithology and oxidisation state. The assigned values were reviewed during the course of the Mineral Resource update and determined to be reasonable based on the supporting dataset. Bulk density values are regarded as being adequate and are supported by historical production data.
Classification	Mineral Resources are stated inclusive of Ore Reserves and depleted to the mined surface as of 31st December 2013. Drill hole density is on a nominal 25m by 25m spacing to the 0 mRL; then on a broader 25m up to 50m average spacing below. Where infill drilling has been completed; will typically correlate to Indicated Mineral Resources. Broader space 50m by 50m drilling typically corresponds with Inferred Mineral Resources.

Criteria	Commentary
	The classification process incorporates a comprehensive and holistic approach of the following criteria with reference to data quality and continuity; − Data type (i.e. hole type, drill hole spacing and orientation, sample type and assay method, etc.) − Statistical performance of the estimate (i.e. kriging efficiency and slope of regression, etc.) − Variography analysis − Estimate parameters (i.e. no. of samples, distance of samples, estimation technique, etc.) − Visual inspections. A combination of the above criteria guides the manual digitising of strings on sections, used to construct envelopes or 3 dimensional solids that are used to control the Mineral Resource categorisation. This process allows a detailed review and assessment of the geological control/confidence of the deposit. The model has been confirmed by successive infill drilling and mine production, which supports the geological interpretation and subsequent classification. The validation of the block model shows a good correlation of the input data to the estimated grades. Historic production and reconciliation performance supports the assigned Mineral Resource classification of Indicated and Inferred Mineral Resources. The Mineral Resource estimate appropriately reflects the view of the Competent Person and is assigned in accordance with the JORC 2012 guideline.
Audits or reviews	Internal peer review by Evolution personnel has been conducted in accordance with Evolution's policy which confirms the stated Mineral Resource and supported the estimation parameters applied. This Mineral Resource has not been audited externally. The process for geological modelling, estimation and reporting of Mineral Resources is industry standard and has been subject to an independent external review. QG undertook a review of the Mineral Resource during October 2013 and found the adopted process to be an industry recognised technique, applied with sensible parameters. Recommendations by QG have been incorporated in this estimate.
Discussion of relative accuracy/ confidence	The relative accuracy of the Mineral Resource estimate is reflected in the reporting of the Mineral Resource as per guidelines of the 2012 JORC Code. The statement relates to global estimates of tonnes and grade. The updated Mineral Resource estimate was compared with production data for the 2013 calendar year. Reconciliation performance for the calendar year was within a +/- 5% tolerance for reported mined and milled tonnes, grade and ounces. The reconciliation result supports the confidence of the reported tonnes, grade, ounces and Mineral Resource classification.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	Commentary
Mineral Resource estimate for conversion to Ore Reserves	The Ore Reserve estimate is based on the current Mineral Resource estimate as described in Section 3. The Mineral Resources reported are inclusive of those Mineral Resources modified to produce the Ore Reserve estimate.
Site Visits	The Competent Person is an employee of Evolution Mining Limited and travels on a regular basis to site. Validation of technical and economic assumptions used in the preparation of this Ore Reserve estimate occurs during these site visits.
Study Status	Mt Rawdon is considered to be a mature operation with over 10 years of historical data. Ore Reserve estimates are generally consistent with current operating practices and experience. On this basis the analysis is considered at a higher level than a Feasibility Study.

Cut-off parameters	Two cut-off grades have been calculated based on the current and forecasted costs and modifying factors, forecast over a period greater than 3 years. These cut-off values are; − Fully Costed – cut-off includes all operating costs associated with the extraction and processing of ore material − Incremental – cut-off grade applies to material that will be mined in the process of gaining access to economic material Ore Reserves are reported a 0.30g/t gold cut-off.
Mining factors or assumptions	The methodology used to convert the Mineral Resource to Ore Reserve can be described as optimisation of existing open pit operations through standard mine planning process steps of pit optimisation, mine design, mine schedule and financial modeling. Factors and assumptions have been formed from existing operating technical assumptions and cost models. On this basis the analysis is considered at a higher than Feasibility Study. Current mining at Mt Rawdon open pit is undertaken via conventional truck and excavator fleet to extract ore material to the ROM, waste material to the waste rock dumps and stockpiling and reclaim of lower grade material. The current mining activities show the appropriateness of this mining method as the basis of the Ore Reserve. Ore dilution and recovery loss is specifically accounted for in the Mineral Resource modeling method and no additional mining dilution or recovery factors are applied to the Mt Rawdon Pit Ore Reserve estimate. This assumption is supported by the actual reconciliation between resource model and mill performance at Mt Rawdon to date being within acceptable uncertainty range for the style of mineralisation under consideration. External and internal Geotechnical studies are carried out to evaluate the operational designs. Ore Reserves are based on the most recent External recommendations of pit slope berm, batter configuration. Inferred material is excluded from the Ore Reserves and treated as waste material, which incurs a mining cost but is not processed and hence does not generate any revenue. The optimisation evaluation shows the ultimate pit size is insensitive to Inferred Resources. The selected mining method does not require additional infrastructure.
Metallurgical factors or assumptions	The ore is to be processed through an existing traditional CIP/ CIL process plant. The current and estimated future average recoveries for gold are 91% and 50% for silver. An operating history of over 10 years supports the metallurgical parameters used in the Ore Reserve estimation.
Environmental factors or assumptions	Mt Rawdon is current with all environmental approvals and compliant to those conditions set out in such approvals.
Infrastructure	The mine is current in operations, thus current infrastructure is adequate to support future operation.
Costs	Capital and operating costs have been determined based on the current operating cost base modified for changing activity levels and reasonable cost base reductions over the life of the mine. On this basis the analysis is considered at a higher level than a Feasibility Study. Site unit costs are applied both as break even site cost used to determine ultimate pit shell and marginal site cost used to define ore waste cut off boundary within the ultimate pit shell. The break even cost base is predicated on similar levels of site activity to recent history with planned cost improvements built in. The marginal cut off cost base is based on the period of low grade stockpile reclaim at the end of mine life. During this reclaim only period mining activity would have ceased and activity level across site would be dramatically reduced relative to current level. No cost impact is expected from deleterious elements and no costs have been included in the Ore Reserve estimate for these. Transport costs and refining charges have been built up from first principles consistent with the application and input assumptions for these costs used by the current operation. State Royalties are 5%.
Revenue factors	Revenue is calculated using a gold price A$1,350/oz and silver price A$22/oz. A typical 3 year trailing average has not been used to set the commodity pricing. Instead a position has been set based on mean broker estimates and the company's longer term view of these commodities.

Market assessment	Gold and silver sold at spot price.
Economic	To demonstrate the Ore Reserve as economic it has been evaluated through a standard financial model. All operating and capital costs as well as revenue factors were included in the financial model. This process has demonstrated that the Ore Reserves for the Mt Rawdon open pit has a positive NPV. Sensitivity was conducted on the key input parameters of cost base, head grade and recovery and found to be robust.
Social	Currently Evolution Mining has agreements with Traditional Owners and on good terms with neighbouring pastoralists.
Other	
Classification	The Ore Reserves are predominantly derived from Indicated Resources. This classification is based on the density of drilling, the orebody experience and the mining method employed. The only Probable Reserves derived from Measured Resources are those reported in known and quantified stockpiles. It is the Competent Person's view that the classifications used for the Ore Reserves are appropriate.
Audits or reviews	This Ore Reserve has not been audited externally.
Discussion of relative accuracy/ confidence	The accuracy of the estimates within this Ore Reserve are mostly determined by the order of accuracy associated with the Mineral Resource model, the metallurgical input and the long term cost adjustment factors used. In the opinion of the Competent Person, the modifying factors and long term cost assumptions used in the Ore Reserve estimate are reasonable.